CIEMAS OPEN CUT SCOPING STUDY

A report prepared for PT. Wilton Wahana Indonesia examining the viability of developing the Ciemas Gold Project by open cut methods followed by underground mining at depth.

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SIGNATURES

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Executive Summary

Introduction

The Ciemas Gold Project is located in West Java, Indonesia, some 160km south of Jakarta. Epithermal gold deposits outcrop and have been delineated to some 150m depth within nine individual deposits. Four of the epithermal deposits have had JORC compliant resources estimated upon them totalling 4.6Mt at 8.4g/t Au (Measured + Indicated + Inferred).

The deposits are hosed by persistent and apparently continuous (vertically and horizontally) fault structures oriented either NW/SE or NE/SW. Mineralization with the fault structures averages 4.0m true width, are sub vertical to some 60 degrees from horizontal.

The project has been in the control of the Wilton Group for some 7 years during which confirmatory drilling, test pit excavation, incline shaft development and several technical studies have been undertaken.

In 2012 the Wilton Group engaged Yantai Design Engineering Co. Ltd (Yantai) to compile a feasibility study to examine the viability of mining the known resources by underground methods. A positive economic outcome was predicted by this study, however restricting the extraction technique to underground means was considered a limiting factor by several reviewers.

The Wilton Group has engaged Mancala Pty Ltd to conduct a Scoping Study to examine the potential economically viability of mining the known resources by open cut methods. The study is to also examine potential underground mining methods following the completion of surface mining. Mancala are to review the technical data set and forward recommendations as to its integrity.

The Wilton Group have expressed a wish for the resources to be developed at a production rate of some 450kta (1,500t/day) for approximately 100,000oz/year of gold production.

Geological Data and Interpretation

The technical data set which the project is based was gathered over some 30 years by various project owners. The historical time span of data collection has resulted in mismatched data with little historical quality control. Drill hole survey data, assayed elements and bulk density data are inconsistent over the collection period.

Only some 17 drill holes have lithological and structural data recorded in the data set.

A detailed search for historical data is recommended along with manipulation of the existing data to refine, update and verify the digital data and geological interpretation.

Data collected in the 2012 drilling program is of high quality and should set the benchmark for historical data manipulation and future drilling programs.

Basic statistical analysis reveals there is no obvious bias in the historical data as compared to the 2012 drilling program. Similar analysis shows that there is no obvious positive relationship between arsenic and gold assays.

Geotechnical data is limited for the project as a whole, especially for near surface areas in the oxidised zone. Based on the geotechnical data collected in the 2012 drilling program (17 holes) the dominant structural orientation consists of steep south to south-west dipping structures. These are evident in all structure types across the three major rock types in the data set. These structures have the potential to contribute to topple style failure on the southern walls of the proposed excavations.

The relationship between defect spacing (RQD) and estimated rock strength has allowed an estimate of the depth from surface that waste material could be ripped with a D8 or similar sized dozer. A depth of 35m has been incorporated into the mine scheduling and cost estimate for "free dig" waste removal.

The ongoing collection of high quality geotechnical data is highly recommended.

Resource Estimate

In June 2014 SRK Consulting reported an updated Mineral Resource Estimate for the Ciemas Project. The SRK work effectively updated a previous estimate made by SRK in 2013 by incorporating all recent drilling data. The SRK 2014 work is reported as compliant to the 2012 JORC Code (Table Ex.1.1-1).

In February 2014 Wilton commissioned PT ASI to prepare a Resources Estimate for the Ciemas Project using a similar data set as the SRK 2014 work. SRK 2014 concluded that in comparing their estimate to that of PT ASI *"there is no considerable discrepancies between the estimates that may result a different conclusion from the Scoping Study".*

Mancala notes that the Resource Estimate will be further updated in the near future incorporating the results from a recent near surface drilling program.

Property	Category	Resource (kt)	Au (g/t)	Au (kg)
Pasir Manggu	Measured	120	7.3	870
	Indicated	450	7.5	3,390
	Inferred	270	3.8	1,030
Cikadu	Indicated	1,100	9.1	9,970
	Inferred	360	8.4	3,040
Sekolah	Indicated	710	9.2	6,520
	Inferred	300	8.6	2,580
Cibatu	Indicated	660	9.1	5,990
	Inferred	670	8.3	5,580
Total	**Measured**	**120**	**7.3**	**870**
	Indicated	**2,920**	**8.9**	**25,870**
	Measured + Indicted	**3,040**	**8.8**	**26,740**
	Inferred	**1,600**	**7.6**	**12,230**

Table Ex.1.1-1 SRK June 2014 Mineral Resource estimate.

HYDROLOGY AND WATER MANAGEMENT

The Ciemas project is located in a tropical monsoon region where annual rainfall is approximately 4,000mm per year. The topographic relief of the area is low, however several apparently significant water courses are present, some of which intersect the proposed open cut excavations.

Significant civil engineering works including four impoundments and some 1.3km of diversion channels (up to 25m deep) are required to re-direct water flows around the open cuts. Knowledge of the peak flow rates within the existing water courses and the ground conditions in which the diversion channels will be excavated is essential. Reliable topographic date (via LIDAR collection) is required for design purposes.

Knowledge of sub-surface hydrology is limited. High flow rates into the open cuts and underground workings from subsurface water sources could seriously impede production.

Regular monitoring of existing bore hole water levels and daily rainfall monitoring is recommended along with draw down and packer tests on selected bore holes.

INFRASTRUCTURE LOCATION AND DESIGN

The conceptual locations of major infrastructure items are depicted in Figure Ex.1.1-1.

Major infrastructure items are considered to be:

- Process plant;
- Mechanical workshop;
- ROM and stockpile area;
- Tailings storage facility;
- Waste dumps;
- Major haul roads;
- Road base quarry;
- Diversion channels and
- Process water dam and other impoundments.

Detailed design and site test work is required for all the civil works to confirm suitable locations and construction materials.

The project will also require detail design work to be completed on:

- Mechanical workshops for both mine and mill with associated component storage and lay down areas;
- Fuel storage and dispensing facilities;
- Analytical laboratory;
- Core/sample storage and sample preparation facilities;
- Communication facilities;
- Training facility;
- Electrical power source, demand and reticulation;
- Messing and accommodation facilities;
- Emergency response and rescue facilities (including fire fighting);
- Medical and security facilities;

- Explosive storage facilities (a magazine of limited capacity is present on site); and
- Technical, supervision and managerial office facilities.

Open Cut Mine design

Open Cut Optimisation

An open cut optimisation exercise has completed to assess the feasibility of open cut mining for the near surface resources on the Ciemas Project. The optimisation used benchmarked operating cost estimates, processing costs and capital cost estimates as agreed with Wilton.

Open Cut Design

The open cut design was completed based on the results of the open cut optimisation study. The design includes four open cuts; Pasir Manggu, Cikadu, Sekolah, and Cibatu (PSM, CKD, SEK & CBT respectively). The physicals of the open cuts are shown in Table Ex.1.1-2 and an image of the excavations and their location are depicted in Figure Ex.1.1-1. The largest open cut, Cikadu, is some 840m in length, has a maximum width of 260m and is 90m deep.

	Ore (t)	Average Au (g/t)	Au (oz)	Waste (t)	Strip Ratio (t:t)	Footprint (Ha)
PSM	250,500	5.8	47,000	3,110,000	12.4	7.4
CKD	1,350,000	6.8	295,000	17,010,000	12.6	18.4
SEK	810,000	6.1	159,000	8,360,000	10.2	13.3
CBT	800,000	6.9	175,000	8,630,000	10.8	12.4
Total	3,210,000	6.6	677,000	37,110,000	11.6	51.5

Table Ex.1.1-2 Open cut physicals. Rounding will cause summation errors.

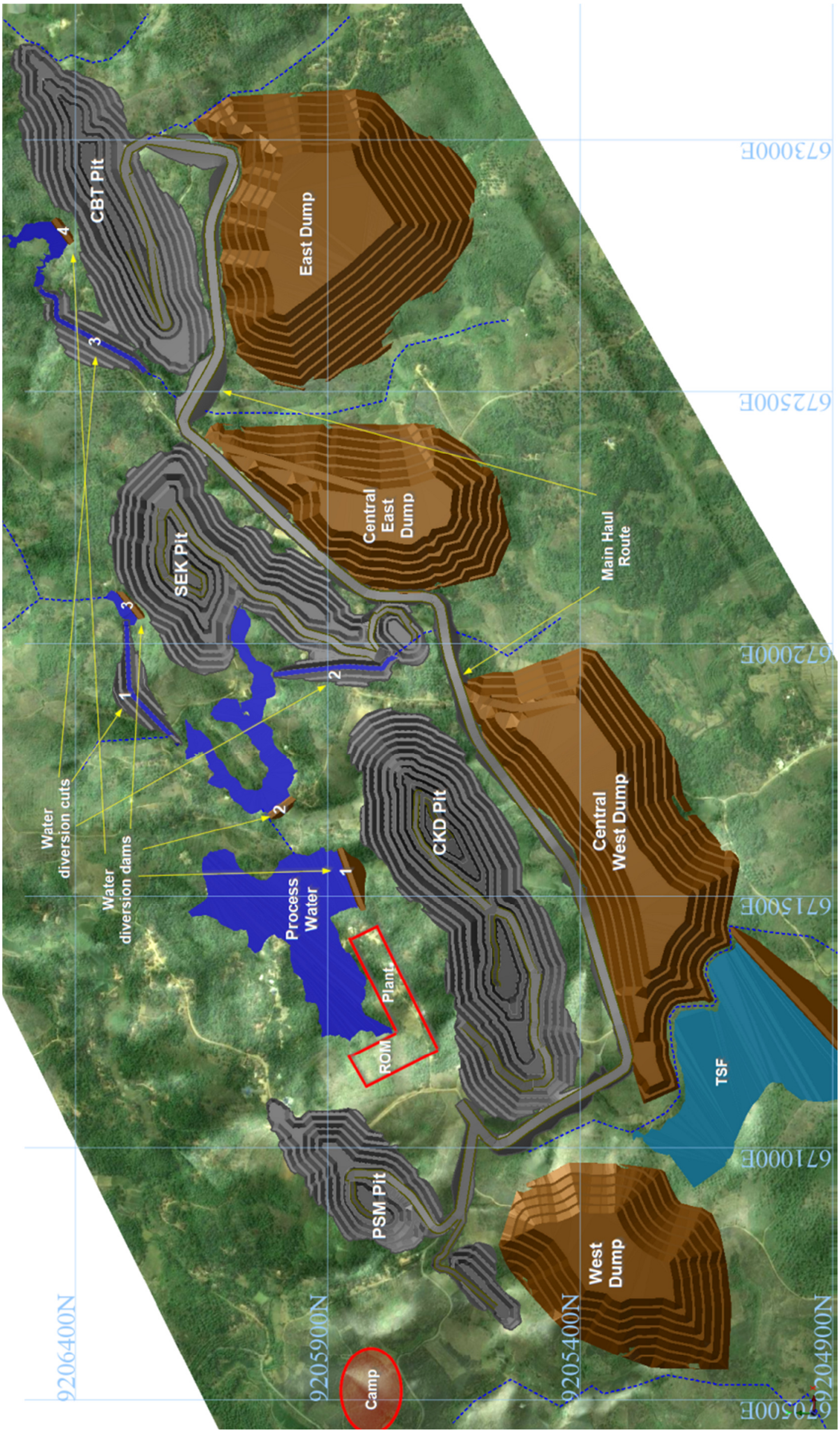

Figure Ex.1.1-1. General site layout.

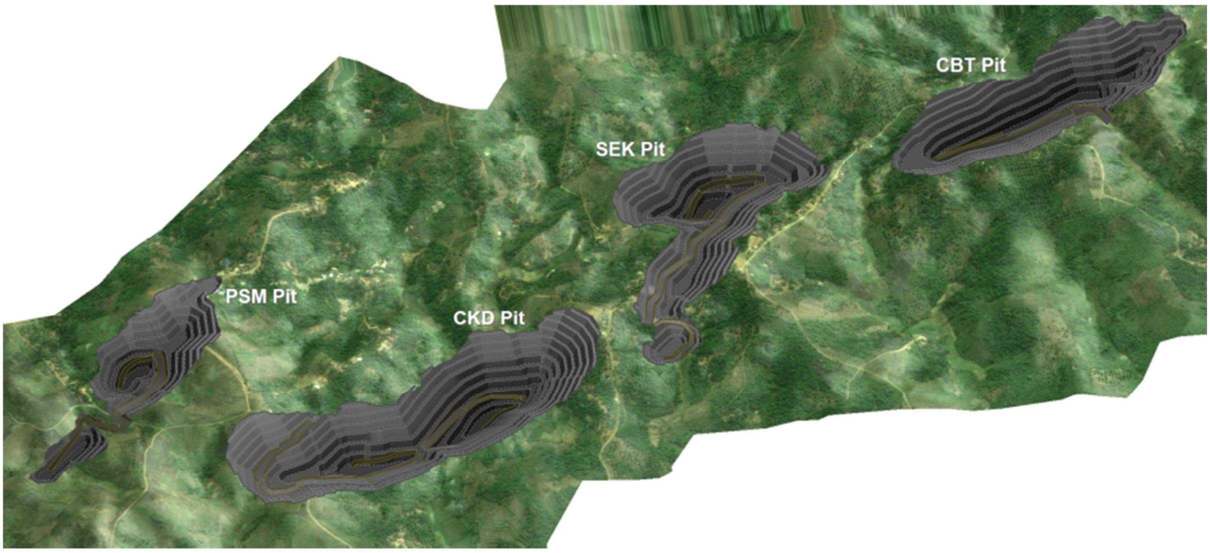

Figure Ex.1.1-2 Isometric view northward of the designed open cuts.

Mineral Resources

The Mineral Resources contained within the open cut designs are shown in Table Ex.1.1-3. 57% of the defined Mineral Resources gold ounces are contained in the open cut mine designs.

The mineral resources as reported in Table Ex.1.1-3 do not contain mining dilution recovery or any other modifying factors.

Resource Class	Tonnes	Au (g/t)	Au (oz)
Total Measured Resources	**100,000**	**8.1**	**23,000**
PSM	100,000	8.1	23,000
Total Indicated Resources	**1,850,000**	**8.5**	**504,000**
PSM	90,000	8.4	23,000
CKD	889,000	8.8	252,000
SEK	460,000	7.9	118,000
CBT	420,000	8.9	119,000
Total Inferred Resources	**670,000**	**8.3**	**178,000**
PSM	10,000	7.7	3,000
CKD	220,000	8.3	59,000
SEK	180,000	8.4	49,000
CBT	250,000	8.2	66,000
Total Resources	**2,620,000**	**8.5**	**712,000**

Table Ex.1.1-3 Mineral Resources contained in the open cut designs. Rounding will cause summation errors.

The Mineral Resources that remain after open cut mining are shown in Table Ex.1.1-4. The remaining resources can potentially be recovered by underground mining methods. The mineral resources as reported in Table Ex.1.1-4 do not contain mining dilution recovery or any other modifying factors.

Resource Class and Lode	Tonnes	Au (g/t)	Au (oz)
Total Measured Resources	**20,000**	**7.7**	**10,000**
PSM Lodes	20,000	7.7	10,000
Total Indicated Resources	**1,070,000**	**9.3**	**320,000**
PSM Lodes	360,000	7.33	86,000
CKD Lodes	210,000	10.0	69,000
SEK Lodes	250,000	11.7	92,000
CBT Lodes	240,000	9.5	73,000
Total Inferred Resources	**930,000**	**7.2**	**220,000**
PSM Lodes	256,000	3.6	30,000
CKD Lodes	140,000	8.7	38,000
SEK Lodes	120,000	9.0	34,000
CBT Lodes	420,000	8.4	113,000
Total Resources	**2,020,000**	**8.3**	**541,000**

Table Ex.1.1-4 Remaining Mineral Resources outside of the open cut designs. Rounding will cause summation errors.

Dump Design

The project is situated in undulating topography with a number of water courses meandering down the side of a hill that drains south eastward. The topography has an impact on the location for waste dumps and water management facilities.

Four waste dumps have been conceptually located south of the open cuts. Their location and access points have been designed with the objective to minimise haul road lengths. (Figure Ex.1.1-1)

Land Disturbance

The total land disturbance from open cut mining will be 168 hectares. Land disturbance by area of individual feature is shown in Table Ex.1.1-5.

Area of Disturbance	Quantity (hectares)
PSM	7.4
CKD	18.4
SEK	13.3
CBT	12.4
Total All Open Cuts	**51.5**
West Dump	14.2
Central West Dump	23.4
Central East Dump	13.8
East Dump	22.7
Total All Dumps	**74.1**
Tailings Storage Facility	10.0
Water Management	11.0
ROM & Plant	6.0
Roads and other	15.0
Total Infrastructure	**42.0**
TOTAL	**168.0**

Table Ex.1.1-5 - Land disturbance by feature.

Mining Schedule

The proposed mine schedule is based on the requirement to produce 1,500 tonnes of ore per day achieving approximately 500,000 tonnes per year. The mine schedule is shown in Table Ex.1.1-6 and graphically on a quarterly basis in Figure Ex.1.1-3. The proposed mine design and schedule is based on the utilisation of two 90 tonne class excavators loading waste and two 45 tonne excavators loading a combination of ore and waste with a fleet of 40 tonne articulated dump trucks.

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Total
Movement (kBCM)	2,140	3,120	3,110	3,130	2,460	1,000	240	15,130
Waste (kt)	5,380	7,800	7,670	7,860	5,960	2,000	440	37,100
Ore (kt)	270	450	600	510	680	480	210	3,210
Au (g/t)	5.8	5.8	6.3	5.9	6.8	7.6	8.3	6.6
Au (koz)	50	83	124	97	149	118	56	677
Stripping Ratio	19.8	17.4	12.6	15.4	8.7	4.2	2.1	11.6

Table Ex.1.1-6- Yearly mine production profile. Rounding will cause summation errors.

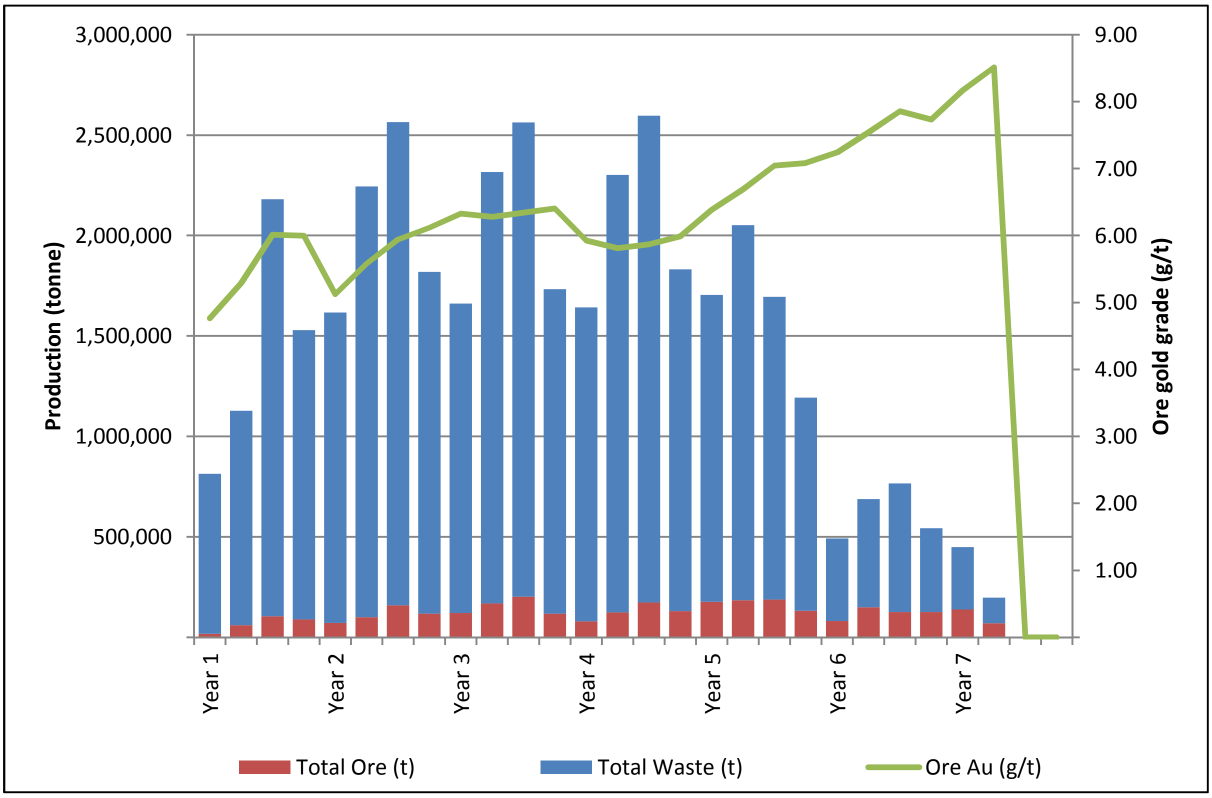

Figure Ex.1.1-3 Quarterly mine production profile.

The production schedule quantities presented in Table Ex.1.1-6 are a combination of the Mining Inventory and Sub-grade ore processed at end of mine life.

The production schedule presented in Table Ex.1.1-6 and Figure Ex.1.1-3 does not constitute an Ore Reserve. The production schedule has been compiled based on low level technical and economic assessments, and is insufficient to support estimation of Ore Reserves or to provide assurance of an economic development case at this stage, or to provide certainty that the conclusions of the Scoping Study will be realised.

Underground Mining

A conceptual mine plan has been developed for mining the remnant mineralisation below and adjacent to open cut workings. Conceptual development and production plans for the Pasir Manggu remnant resources have been constructed.

The preferred mining methods are:

- Residual ore below the floor of the open cut: Overhand Benching with Fill;
- Remnant ore adjacent to the sides of the open cut: Underhand Benching with Pillars.

Both methods are mechanised and require access and ventilation sites developed within the open cuts.

At Ciemas ore zones can be up to 600 metres in length, have an average width of 4 metres and a bulk density of 2.7, this equates to a 6,500 tonnes per vertical metre. However, the underground mineable (high grade portion) is likely to be in the order of 50 to 60% of this number, that is around 3,000 t/vm. Typical production rates from narrow vein mines of this tenor are around 250,000 tonnes per annum. To achieve the planned production rate (1,500tpd) at least two underground ore sources are required to be concurrently mined.

Following open cut mining the remaining resource potentially exploitable by underground methods amounts to: **2.02Mt at 8.3g/t Au for 541k oz of gold (at 1.0g/t cut off)**

Financial Evaluation

A financial model for the surface mining portion of the project has been developed using prior (SRK report) non mining related capital and operating costs and benchmarked mining capital and operating costs. The results of the evaluation are summarised in Table Ex1.1-8.

Conceptual capital costs for the underground mining are presented in Table Ex1.1-7.

Development Area	Metres	$/metre	Total $
Portal	15	10,000	150,000
Decline 1:7	350	4,000	1,400,000
Level – waste capital	200	3,500	700,000
Ventilation	50	3,000	150,000
Escape way	n/a	4,000	200,000
Pumps/Vent Fans etc.			900,000
Total			3,500,000

Table Ex1.1-7 Estimated capital costs associated with an individual mining operation. (Exclusive of capital associated with mobile mining equipment)

Typical operating costs for narrow vein mining operation are forecast to be in the region of $70 - $80 per tonne of ore mined.

Mine Physicals	Value	Comment
Mine Life	7 Years (Open Cut)	Potential 3-4 years additional underground mining
Mining Method	Free dig and subsequent drill, blast load and haul	Some 75% of all waste is estimated as free dig
Production Profile	3,210,000 tonnes at 6.6g/t	For 677,000 oz Au
Average Stripping Ratio	11.6	Tonne:Tonne
Average Waste Movement/year	5,290,000 tonnes	
Maximum Waste Movement/year	7,860,000 tonnes	Year 4
Average Ore Production/year	460,000 tonnes	
Maximum Ore Production/year	680,000 tonnes	Year 5
Direct feed Ore Cut Off Grade	3.0g/t Au	
Sub-Grade Ore	1.0 – 3.0g/t Au	Stockpiled for processing in Year 7
Average dilution	16%	Variable block by block basis
Average recovery	95%	Applied globally
Owner operator/contractor	Owner Operator	Mobile capital equipment included in costing
Mobile plant	2 x 90 tonne + 2 x 45 tonne excavators and maximum 10 x 40 tonne ADT's	Plus ancillary equipment
Nominal Processing Rate	500,000 tonnes per year	
Processing Recovery	90% of plant feed	Silver credits not considered
Financials	Value	Comment
Pre-Production CAPEX	$86,000k	Includes mobile surface mining plant.
Mining Cost	$3.45/ tonne material mined	Includes pre-production capitalised OP costs
Mining Cost	$175/oz	Includes pre-production capitalised OP costs
$NPV_{(8)}$ (Post Tax Ungeared)	$186,000k	Discount Rate 8%. CIT assumed at 25%
IRR (Post Tax)	53%	
C1 Cash Cost	$451/oz Au	
C2 Cash Cost	$584/oz Au	
C3 Cash Cost	$633/oz Au	
Project Cash Flow (EBITDA)	$488,000k	
Gold Price	$1,300/oz Au	Applied through project life

Table Ex1.1-8 Physical and financial summary surface mining at Ciemas.

The physical and financial outcomes presented in Table Ex1.1-8 have been estimated from low level technical and economic data, which are insufficient to support the estimation of Ore Reserves, or to provide certainty that the conclusions of the Scoping Study will be realised.

It is recommended that the financial model be refined with the object of increasing the estimated NPV by:

- Incorporating cost data based on supplier quotations (consumables etc.);
- The inclusion of labour options including a mixture of expatriate trainers and national operators;
- Incorporate a contractor option, which would reduce upfront capital and improve NPV;
- Refine the mine schedule to reduce waste stripping and pre-production earth works;
- Incorporate a variance of productivity over time which may reflect ramp up activities and the gradual replacement of expatriates by national labour;
- Civil and mechanical engineering designs delivering material lists for supplier quotations;
- Contractor quotations of construction activity for comparative purposes;
- Updated capital and operating costs for the process plant;
- Refined revenue forecasts based on updated metallurgical testing, and refining quotations;
- The inclusion of capital funding mechanisms, which if include debt, the cost of which is accounted for; and
- The definition of and inclusion of an operational and management agreement with the preferred supplier to manage and operate the mine.

NEAR SURFACE UNDERGROUND VS. OPEN CUT MINING METHODS

The SRK Report calculates a project NPV at a 10% discount rate using the mine design presented by Yantai. In its calculation, SKR has used various assumptions which differ to those of Mancala. To permit a realistic comparison of the two mining methods physical and financial outcomes and the major perceived risks, Mancala has partially modified the SRK modelling assumptions. Mancala has used these modified assumptions to estimate the financial outcome of the Yantai mine design and compared it to the Open Cut design Table Ex1-1.1-9

The most significant modification made by Mancala to the SRK/Yantai assumptions is to the estimated mining costs. SRK/Yantai estimate mining costs of $22.60/ore tonne. Based on bench marking and recent experience, Mancala estimate the mining cost for a mechanised, underground, narrow vein mining operation in Indonesia would be in the region of $80/ore tonne.

SRK/Yantai have assumed depreciation of the initial capital cost (USD 93M) at $7.107M/year, resulting in some $50M not being recouped over the project life. Mancala's open cut depreciation is also straight line, but is totally recouped over the project life. An adjustment to the SRK/Yantai costs has been estimated by Mancala to account for full capital payback over the project life (additional $8.3M/year in costs).

Mancala's NPV is reported after CIT at 25%. Mancala has made an adjustment to the SRK reported NPV to account for CIT.

SRK estimate of NPV is based upon a gold price of $1,400/oz while Mancala's work is based upon $1,300/oz. The SRK NPV has been adjusted by revising down revenue based on a gold price $1,300/oz.

The discount factor to NPV in Mancala's work is 8%, while the SRK model uses 10%. No adjustment has been made in this regard by Mancala.

Mancala's adjustment to the SRK financial model are estimates based upon the reported outcomes. Further accuracy would be gained if the assumptions were incorporated into the SRK base financial model. Mancala is not privy to this model.

Mine Physicals	Open Cut	Underground	Comment
Mine Life	7 Years	6 Years	3-4 years U/G mining after open cut
Production Profile	3.2 Mt 6.6g/t	2.44 Mt 7.1g/t	480k Au oz of Resource remaining after O/C
Gold (oz)	677,000	557,000	
Cut Off Grade	1.00	1.69	For O/C COG easily varied over mine life
Average mining dilution	16%	17%	
Ave. mining recovery	95%	85%	
Processing Rate (tpa)	500,000	450,000	
Processing Recovery	90%	90%	
Financial Outcome			
Pre-Production CAPEX	$86,000k	$93,000k	Owner operator O/C mining fleet, potential reduction if contractor option considered.
Pre-production schedule	1 Year	2 years	For O/C, further reduction in time frame possible with advanced scheduling
Operating Cost per ore tonne	$90	$123	For U/G, increase in mining cost of $57.20/ore tonne with respect to SRK Report. Based on benchmarking.
Gold Price (USD/oz)	$1,300	$1,300	Decreased from $1,400 for U/G
Project Cash Flow (EBITDA)	$488 Million	$315 Million	Decrease from $517M with adjusted Au price, increase mine costs and depreciation.
Discount rate	8%	10%	
NPV(8) (Post Tax Ungeared)	$186 Million	$120 Million	For U/G decrease from $210M with increase mining cost and post-tax.
Perceived Risk			
Availability of Miners	Low	Moderate - High	75 skilled miners required for U/G.
Availability of mining plant and serviceability	Low	Moderate - High	Remote area - limited OEM services available. Specialised U/G mining equipment.
Impact of poor ground	Low	High	Very poor ground conditions near surface.
Impact of water	Moderate	High	High rain fall and high water table.
Resource recovered	Low	Moderate	Pillars and unrecovered ore in U/G, suspected undefined resources recovered in O/C.
Surface impact	High	Low	Large footprint for waste dumps and open cuts.

Table Ex1-1.1-9 Comparison of open cut (O/C) mining methodology and initial underground (U/G) works. U/G financial outcome modified from SRK Report.

CONCLUSIONS AND RECOMMENDATIONS

The following conclusions were reached:

- Open cut mining of the deposit based on the optimised pits provides a better financial outcome compared with underground mining of the upper zones of the deposit;
- Adopting the open cut mining method increases the gold recovery per vertical meter as no pillars are left behind for support;
- Adjacent, un-minable (by underground) mineralised lenses will be recovered by the open cuts;
- Adverse ground conditions and ground water control are better managed by an open cut compared with underground methods;
- Mining risks are significantly reduced using the open cut method; and
- Open cut mining significantly increases the area of land disturbance compared with underground mining.

The following recommendations are made:

- Development of the Ciemas Project should be changed to incorporate open cut mining of the upper zones of the deposits. The depth of the open cuts will be determined by optimisation techniques and comparative analysis of underground mining costs with waste stripping costs.
- Investigate the land usage and social impact for the open cut mining option.

In general terms, a feasibility study is required for the project which is aimed at forecasting actual cost (capital and operating), site productivity and revenue parameters to a level of +/- 15% of actual. Such documentation when, independently reviewed would form a basis upon which project financiers or equity contributors could assess the technical, social, political, environmental and financial risk they may be exposed to.

In a broad sense, the high grade, near surface, apparently conventional metallurgically treatable ore would suggest the project is robust.

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1 Background and Scope

1.1 Background

On the 6th February 2014, PT Wilton Wahana Indonesia (PT WWI) commissioned Mancala Pty Ltd to review the Ciemas Project data set with the aim of providing a conceptual mine plan involving open cut mining followed by potential underground mining at the base of the competed open cuts.

PT WWI is a wholly owned subsidiary of the Singapore Stock Exchange listed Wilton Resources Corporation (Company Registration Number 20030095D) and is the 100% beneficial holder of the Mining Business Licence (Izin Usaha Pertambangan) 503.8/7797-bppt/2011. PT WWI also own 95% of the shareholding of PT Like Tucha Ciemas, the holder of Mining Business Licence 503.8/3106-bppt/2012. The two business licences cover the area known as the Ciemas Project.

Wilton Resources Corporation publically stated business model is “business of exploration, mining and production of gold doré in Indonesia”.

Mancala Pty Ltd (Australian Business Number (ABN) 65 009 579 560) is a member of the Mancala Group of companies, domiciled in Australia with the ultimate Australian Holding Company being Mancala Holdings (ABN 55 009 587 679). Mancala Holdings is 100% owned by the Singapore Stock Exchange listed Sapphire Corporation Limited (Company Registration Number 198502465W).

The Mancala Group’s business involves the provision of specialist mining services, which includes mine management, mine consultancy contract mining, raised boring, shaft excavation, engineering services and other mining services. The company has been operating within the mining services market for 24 years. Although the company’s operations have been concentrated within Australia, international work has been undertaken in Botswana, Fiji, Papua New Guinea, Indonesia, New Zealand and Vietnam.

The Mancala group is well regarded amongst its peers as a specialist narrow vein miner, with technical skill, appropriate equipment and managerial ability to extract high grade resources safely with minimal ore loss.

1.2 Scope of Work

Following a visit to the Ciemas Project site, discussions between Mancala and PT WWI resulted in PT WWI engaging Mancala via a consultancy agreement to examine an existing project data set and to advise as to the viability of mining the gold deposits via initially open cut methods followed by underground extraction of deeper resources.

Mancala proposed a multi-stage approach to the mining study, with Stage 1 the conceptual high level assessment reported herein. The results of Mancala's work herein rely exclusively on other professional's data the veracity of which is commented upon along with recommendations.

The agreed scope was to:

- Preform a Whittle Four-X open cut optimization on the individual resources;
- Run an optimization profile on 2, 3, and 4 pits operating simultaneously;
- Select the optimum open cut design(s) and modify it manually to incorporate access ramps, and localized modification to batter/berm designer (the basic operational open cut design);
- Assess the open cut shells which are deeper than optimum in an attempt to determine the cost/benefit of mining the areas of poor ground conditions with surface techniques rather the expensive underground techniques;
- Provide an infrastructure (dump(s), haul roads, process plant, tailings dam) location and conceptual design which would allow extraction of existing surface resources, the underground resources and the impact of any future exploration success;
- Present a mining inventory estimate based on assumed resource modification factors;
- Present a production schedule (monthly basis) reporting tonnage of ore/waste and contained gold etc.;
- Present a cost/revenue forecast and a sensitivity analysis based upon major costs/revenue parameters;
- Conduct a bench marking exercise with similar operations;
- Provide an assessment of the individual resource areas to determine optimum production profiles and recommend whether multiple underground operations are required simultaneously to meet the preferred production rate (450kta);
- Assess the timing for commencement of underground operations to allow uninterrupted production during the transition from surface to underground operations;
- Provide a conceptual access design which would incorporate portal(s) location, decline development and ventilation shaft locations;
- Preform a geotechnical review and assess the implications as to mining method selection;
- Recommend mining methods for the various geotechnical domains (if apparent);
- Provide an indicative site layout which will permit the chosen mining methods to be carried out;
- Indicative (based on bench marking) capital, operating and associated costs; and
- Indicative equipment selection and its capital cost.

1.3 Sources of Data and Information

Over the period 1st to 5th of November 2013, staff of PT WWI hosted Messrs Tim Akerman and Bill Lannen (both Executive Directors of Mancala) at their Jakarta office and on an inspection of the Ciemas Site.

During the office visit, commercial and technical discussions were undertaken along with a preliminary review the projects technical data set.

Some six hours were spent at the Ciemas site inspecting the current exposures at the southern end of the Pasir Manggu ore zone and drill core from all four resource zones. The core was systemically examined by referencing drill sections in relation to:

- Ground condition variation with depth and resource zone;
- Presence of free gold within resource zones;
- Subjective correlation of quartz vein intensity and Au grade;
- Presence or absence of sulphide mineralisation vs. depth of intersection;
- Hanging wall and footwall contact style of the resource zones (i.e. planar or irregular, visually determinable etc.; and
- Presence or absence (visual) of disseminated mineralisation beyond the identified resource zones.

In addition to the on ground examination of the Pasir Manggu outcrop, a potential quartz porphyry gold bearing zone (Cipirit) was also visited. The area shows evidence of extensive artisanal mining activities. This area and its potential resources are not within the scope of the work herein.

Digital Data requested by Mancala and provided by PT WWI is presented in Table 1.3-1.

Data Format	Data Type	File Name	Description
Digital	Topography	PSMW-CIKADU-SEKOLAH-CIBATU_OxideSurface.dtm	Digital Terrain Model : Oxide Surface
Digital	Topography	PSMW-CIKADU-SEKOLAH-CIBATU_OxideSurface.str	Surpac String File Model : Oxide Surface
Digital	Topography	PSMW-CIKADU-SEKOLAH-CIBATU_SoilSurface.dtm	Digital Terrain Model : Soil Surface
Digital	Topography	PSMW-CIKADU-SEKOLAH-CIBATU_SoilSurface.str	Surpac String File Model: Soil Surface
Digital	Topography	PSMW-CIKADU-SEKOLAH-CIBATU_TopoSurface.dtm	Digital Terrain Model: Surface Topography
Digital	Topography	PSMW-CIKADU-SEKOLAH-CIBATU_TopoSurface.str	Surpac String File Model: Surface Topography
Digital	Drill Holes	DH_ASSAY.csv	Drill Hole and Costean Assays. 6,794 records
Digital	Drill Holes	DH_COLLAR.csv	Drill Hole and Costean Collar Details. 606 records
Digital	Drill Holes	DH_DSURVEY.csv	Down Hole Survey Data. 822 records
Digital	Drill Holes	DH_LITHOLOGY.csv	Drill Hole Lithology Data. 1,103 records. 17 Holes
Digital	Drill Holes	DH_ASSAY.DAT	Drill Hole and Costean Assays. 6,794 records
Digital	Drill Holes	DH_COLLAR.DAT	Drill Hole and Costean Collar Details. 606 records
Digital	Drill Holes	DH_DSURVEY.DAT	Down Hole Survey Data. 822 records
Digital	Drill Holes	DH_LITHOLOGY.DAT	Drill Hole Lithology Data. 1,103 records. 17 Holes
Digital	Block Model	BM_CIBATU_OK_20140104.csv	Cibatu block model data. 73,115 records
Digital	Block Model	BM_CIKADU_OK_20140104.csv	Cikadu block model data. 77,775 records
Digital	Block Model	BM_PSM_EXT_OK_20140104.csv	Pasir Manggu block model data. 11,645 records
Digital	Block Model	BM_PSM_WN_OK_20140104.csv	Pasir Manggu block model data. 6,473 records.
Digital	Block Model	BM_PSM_WS_OK_20140104.csv	Pasir Manggu block model data. 37,410 records
Digital	Block Model	BM_SEKOLAH_N_OK_20140104.csv	Sekolah block model data. 35,187 records
Digital	Block Model	BM_SEKOLAH_S_OK_20140104.csv	Sekolah block model data. 19,235 records
Digital	Block Model	Various	17 Micromine block model files. Not used in Mancala's work.
Digital	Bock Model	Block Definitions.xlsx	Spread sheet defining the block model attributes
Digital	Wireframes	PSM1 – PSM11 dfx, str and dtm	Wireframes in various formats of the Pasir Manggu ore zone
Digital	Wireframes	CKD12 – CKD19 dfx, str and dtm	Wireframes in various formats of the Cikadu ore zone
Digital	Wireframes	SEK20 – SEK30 dfx, str and dtm	Wireframes in various formats of the Sekolah ore zone
Digital	Wireframes	CBT31 – CBT35 dfx, str and dtm	Wireframes in various formats of the Cibatu ore zone
Digital	Infrastructure	CIBATU-SEKOLAH Infrastructure Map.dwg	As file name
Digital	Infrastructure	CIB-SEK 5-System legend.dwg	Not used in Mancala's work
Digital	Infrastructure	CIB-SEK 400 level layout drawing.dwg	As file name. Not used in Mancala's Work
Digital	Infrastructure	CIB-SEK 440 level layout drawing.dwg	As file name. Not used in Mancala's Work

Digital	Infrastructure	CIB-SEK General plan.dwg	Topography, ore zone intersection with topography and proposed infrastructure (earlier study)
Digital	Infrastructure	CIB-SEK Geological-Topographic Map.dwg	As file name
Digital	Infrastructure	CIB-SEK Mining method legend.dwg	Shrink stope mining method. Not used in Mancala's work
Digital	Infrastructure	CIB-SEK Shaft and Roadway section.dwg	As file name. Not used in Mancala's work
Digital	Infrastructure	CIKADU-Infrastructure Map.dwg	As file name.
Digital	Infrastructure	Development planning (latest) Pasir Manggu - arranged plan (dual deviated + adit + mill).dwg	As file name.
Digital	Mine Design	Ciemas Mining Design (Folder)	Series of plans depicting the earlier Shrink stope mining design. Not used in Mancala work
Digital	Mine Design	Pasir Manggu Tunnels (Folder)	Series of plans depicting the earlier Shrink stope mining design. Not used in Mancala work
Report	Hydrogeology	HYDROGEOLOGICAL REPORT OF CIEMAS PROJECT- FINAL.pdf	As file name. Describes lithological occurrences in a number of bore holes. No laboratory or field testing completed.
Report	Geotechnical	Geology & Geomechanic Report Of Ciemas -Final_Complete.pdf	As file name. Field and laboratory testing of representative rock units. Limited discussion of results
Report	Survey	Final Report-Eng.pdf	Bench mark survey of 9 survey points on Ciemas area
Report	Metallurgical	FINAL REPORT PT ASI 2013.pdf	Report on test work of samples from Pasir Manggu. Gravity, and CIL test work completed on relatively low grade samples. Combination gravity and CIL recommended.
Report	Resource Estimate	Final Resource estimation report ASI WWI Jan 2014 REV1.pdf	Recent (2014) update of mineral resources at Ciemas. "The ASI Report"
Report	Sections	Drill Sections (Parry) Folder	99 drill sections and various spacing's depicting the geological interpretations and intersection grades. Sections date from 1993.
Report	Drill holes	Geology and Geotech Logging DDH 1022_edit ASI.xls + 14 others similarly named.	Detailed geological and geotechnical logging of 15 drill holes. DDH 1022, 1023, 1025, 1026, 1036, 1037, 1042, 1124, 1131, 1132, 1138, 1141, 1142, 1143, 1144.
Report	Drill holes	SEKOLAH_DDH1022.PNG + 14 others similarly named	Cross section of the logged drill holes showing lithology, grade and historical drill holes.
Report	General	Geological Evolution Report on the Ciemas Gold Mine, Indonesia. 2010	Publically available from the Wilton Website
Report	General	Circular to Shareholders by Hatawan (subsequently Wilton) dated 26 September 2013	Publically available from the Wilton Website. Contains the Independent Qualified Person's report ("The SRK Report" and the Independent Valuation Report ("The GCA Report")
Report	Resource Estimate	Updated Resource Report for the Ciemas Gold Project, SCN398	Recent (2014) JORC 2014 update of mineral resources at Ciemas. By SRK ("The 2014 SRK Report")

Table 1.3-1 Data provided by PT WWI to Mancala.

1.4 Units and Currency

Metric units are used throughout this report unless noted otherwise. Currency is United States dollars ("US$" or "$"), Australian dollars ("A$"), or currency as specifically stated. Where required, an exchange rate between US$ to A$ of 1:0.93 has been used. For converting grams of gold to ounces of gold, a factor of 31.1035 grams per troy ounce is used.

2 RELIANCE ON OTHER EXPERTS

This report has been compiled by Mancala staff under the supervision of Mr Tim Akerman, Executive Director of the Mancala Group. The Mancala personnel are experienced technical professionals in their respective areas of expertise.

Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false or misleading at the date of this Report.

The following people have contributed to this report, and have done so under the supervision of Tim Akerman, who has also written, edited and reviewed sections of this report. Their areas of expertise and section contributions are listed below:

<u>Tim Akerman</u>

Qualifications: BSc. (Hons.) Geology

Affiliations: Member of AusIMM and Member of the Australian Institute of Company Directors

Experience: 28 years

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Sections: 1, 2, 3, 4, 5, 6, 7, 17, 19, 20, 21

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<u>Adrian Molinia</u>

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Experience: 42 years

Position: Principle Mining Engineer

Sections: 1, 7, 10, 11, 17, 18,

3 Introduction

The Ciemas Gold Project is located in West Java, Indonesia, some 200km south of Jakarta. Epithermal gold deposits outcrop and have been delineated to some 150m depth within nine individual deposits. Four of the epithermal deposits have had JORC (2012) compliant resources estimated upon them totalling some 4.64Mt at 8.4g/t Au (Measured + Indicated + Inferred).

3.1 Project Area

The deposits are hosed by persistent and apparently continuous (vertically and horizontally) fault structures oriented either NW/SE or NE/SW. Mineralization within the fault structures averages 4.0m true width and is sub vertical to some 60 degrees from horizontal.

The project has been in the control of the PT WWI for some 6 years, during which confirmatory drilling, test pit excavation, incline shaft development and technical studies have been undertaken.

PT WWI has expressed a wish for the resources to be developed at a production rate of some 450kta (1,500t/day) for approximately 100,000oz/year of gold production.

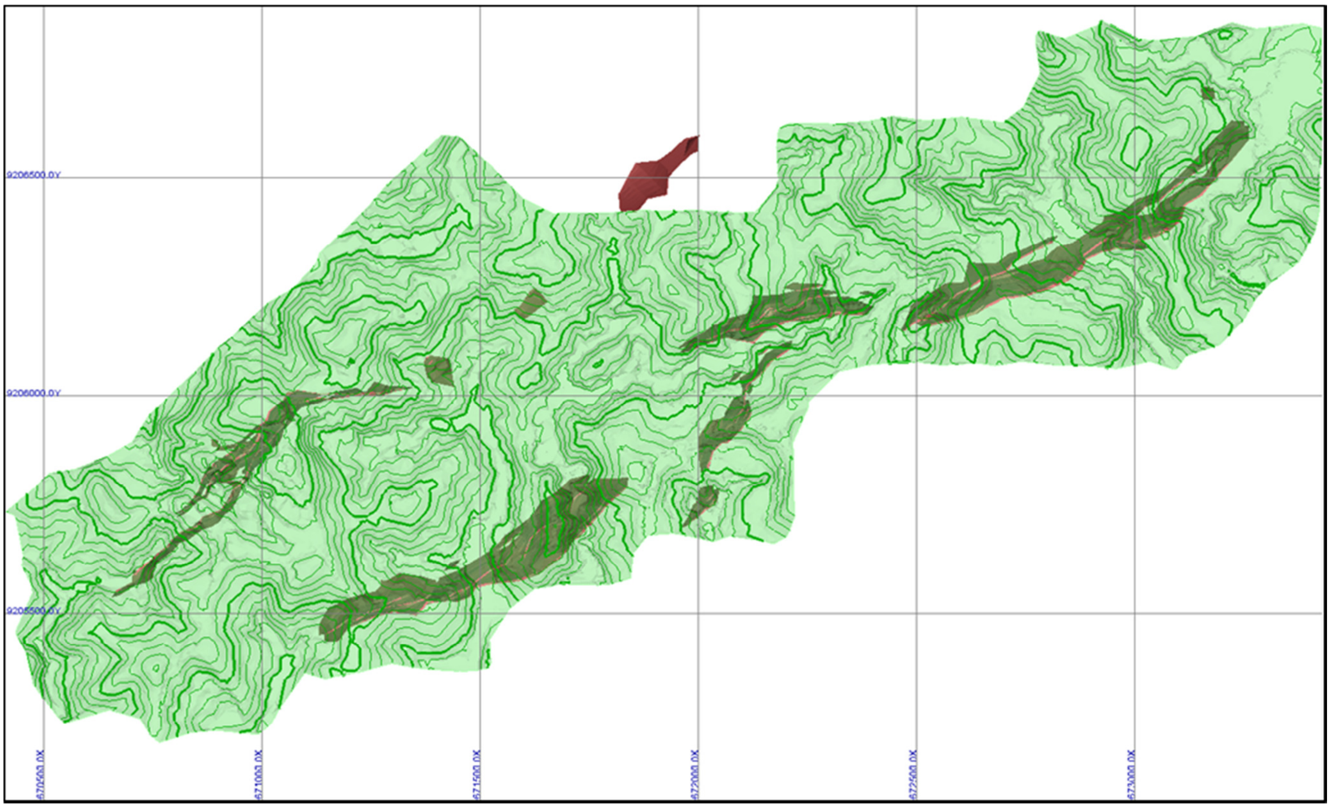


Figure 3.1-1 Plan view of study area with topography (5.0m contours) and resource areas projected to surface. Grid lines at 500m spacing.

4 Property Description and Location

The following text is taken directly from the "SRK Report" and is included for the reader who has not had the benefit of its reference.

4.1 Regional Location and Access

Administratively, the Ciemas deposit area is located in the Jampang Kulon area, in the south-western part of the Sukabumi Region, West Java Province, Republic of Indonesia, 200 km south of Jakarta.

An expressway connects Jakarta and the city of Bogor (55 km), from where a secondary paved road leads through Sukabumi to the coastal city of Pelabuhan Ratu, from where access to the mine and exploration area is provided by 45 km of paved asphalt road. Generally, access to the area is convenient. However, the road deteriorates as it approaches the mine. Figure 4.1-1 shows the regional and local location of the project area.

Figure 4.1-1 Regional location of Ciemas Project area.

4.2 Topography and Climate

The landform of the exploration and mining area is represented by undulating terrain with elevations varying from 379 to 760 m above sea level ("ASL"), generally with the lower parts in the southern areas.

The typical monsoon tropical climate is characteristic of the West Java province; the year has two seasons, dry and rainy. The temperature is stable year round, remaining between 18° and 28℃ day and night. Precipitation is nearly 4,000 mm per annum, mostly concentrated between November and April, which is the rainy season.

Water resources are abundant and the level of groundwater is high. Most of the ore bodies are located below the groundwater table. Sukabumi has a tropical monsoon climate, with hot weather, thick soil layers, and dense vegetation.

4.3 Regional Economy and Infrastructure

The project is located in an impoverished mountainous area. The local economy is based mainly on agriculture. Main crops include rice, bananas, corn, and papayas. Plantations of cloves, rubber, and tea are also common.

Presently the power supply is via the local grid; generators are another major source of electricity. A large-scale power station and port project are under construction in Pelabuhan Ratu, about 17 km in a straight line from the mine site.

The water supply is sufficient due to the extremely well-developed river system and high levels of precipitation; water pools and elevated tanks are available on the mine site.

Wilton is one of the few mining enterprises in the Ciemas area; in some places local people pan gold from strongly altered volcanic rock outcrops and soils.

The Indonesian government is focused on attracting investment and increasing employment opportunities. Wilton intends to recruit a majority of project employees from the local population.

5 History and Local Geology

The following text (Sections 5.1 and 5.2) is taken directly from the "ASI Report" and is included for the reader who has not had the benefit of its reference.

5.1 History

Gold was already known to be mined at Ciemas back during the colonial times. Since then it appears that local mining has also occurred over many decades. Madam Like Tucha was the first to acquire a KP (Kuasa Pertambangan) over the Pasir Manggu area in 1985.

In 1986, a former Australian company, Parry Corporation Limited ("Parry"), contracted with Like Tucha (the concession holder at the time) and commenced exploration work in the project area. Detailed exploration work was concentrated in Pasir Manggu, consisting of geological mapping, geochemical and geophysical surveys, extensive outcrop sampling, trenching (called "costean" by Parry), pitting, reverse-circulation ("RC") drilling, and diamond drilling. Diamond and RC drilling, as well as pit sampling and trenching, were also conducted in the deposit areas of Cibatu, Cikadu, and Sekolah. Most of the diamond drill holes ("DDH") conducted in the Project were completed by Parry between 1986 and 1990.

Another Australian company, Terrex Resources NL ("Terrex"), joined the exploration from 1990 to 1994. Work carried out by Terrex included RC drilling, percussion drilling, and some trenching (costean). The exploration was focused on the targets of Pasir Manggu, Cibatu, Cikadu, and Sekolah; and resources in these areas were preliminarily estimated based on extensive sample results. During this time, Terrex started prospecting on other deposits in the project area. An Australian-Indonesian joint venture, PT. Meekatharra Minerals ("Meekatharra"), conducted a detailed follow-up exploration in the project area from 1995 to 2000. Meekatharra reviewed and evaluated previous geological data, and additional exploration completed during this period included detailed geological mapping and additional sampling from trenches and pits, as well as diamond drilling evaluation. In the Ciaro porphyry copper-gold deposit area, a total of eight additional holes were drilled to further the geochemical and geophysical prospecting.

PT WWI entered the project in 2008 by acquisition of the Ciemas Gold Project. Geophysical prospecting including Induced Polarization ("IP") and a ground magnetic survey was conducted across the Pasir Manggu quartz veins in 2008. Wilton also completed some trenching and pitting as well as surface sampling in the Project.

Of all the deposits, Pasir Manggu is considered the most extensive in terms of exploration, followed by Cibatu, Cikadu, and Sekolah. Feasibility study reports were prepared for the Pasir Manggu deposit in 1997 (by Meekatharra) and 2012 for WWI by ShanDong Gold Group, Yantai Design Research Engineering Co., Ltd. This was based around the concept of underground development of the deposits. The outcome of the preliminary study was positive, and WWI embarked on a process to take a holding company public on the Singapore Stock Exchange.

WWI commissioned SRK Beijing to produce an independent qualified persons report (IQPR) on the project, including a Resource assessment compliant with the 2004 JORC Code, as well as a review of the development proposals. This formed the basis of a financial evaluation of the project by Greater China Appraisal (GCA). These documents together formed the basis of the technical documentation incorporated in the RTO documentation that resulted in the acquisition of Hartawan Holdings Ltd, an SGX- Listed counter, which has been renamed Wilton Resources Corporation Limited (WRC).

In combination with the SRK evaluation, Wilton completed a total of 24 diamond drilling holes (DDH) to verify the historical data and explore the gold mineralization at Pasir Manggu West, Cikadu, Sekolah, and Cibatu. Core samples were prepared by the Intertek Laboratory in Jakarta and were analysed with fire assays.

To date, the major exploration work completed in the Ciemas Gold Project area consists of detailed geological and topographical mapping, geophysical and geochemical surveys, 360 costean/trenches/pits, 216 DDHs, 115 RC drill holes (reverse circulation hole or RCH), 7,500 hand auger drill holes, and 120 percussion drill holes. In addition, preliminary metallurgical engineering and geotechnical studies have been completed, along with small pilot mining studies. This report utilises the recent additional drilling performed by WWI to provide a basis for a JORC-compliant revision of the Resource base of the Ciemas Project. Most attention has been paid to refining the database of survey and sample information, the modelling of database, the refining of geostatistical parameters, and a re- estimation of the resource based upon newly generated block models utilising the more refined wireframes of mineralisation. This work has enabled significant positive revision to the Resource classifications which are reported in here.

5.2 Local Geology

The Ciemas Gold Project is situated within a volcanic polymetallic, metallogenic belt in Ciletah Bay, Indonesia, containing gold (“Au”), silver (“Ag”), lead (“Pb”), zinc (“Zn”), and copper (“Cu”). The lithology of the belt is formed mainly of volcanic breccia and mostly covered by Quaternary eluvium and alluvium as well as a post-mineralisation tuff blanket up to 20 m thick. Volcanic breccia, tuffs, and andesite are widely distributed in the Project area.

Gold mineralization in Ciemas is hosted in quartz veins or structurally altered rocks with tectonic breccia, or in quartz dacite porphyry. Mineralisation is predominantly related to NW-SE and E-W veins with the extensions varying from 100 to 900 m; the width of the ore bodies are generally 2 – 14 m. Most gold mineralized bodies present in the northeast zone contain brecciated chalcedony-quartz carrying pyrite, arsenopyrite, and small amount of galena and sphalerite mineralization. The zone is contained in strongly silicified clay (argillic alteration) several metres wide containing disseminated pyrite. The indistinct external propylitic alternation envelope features chlorite and scattered pyrite.

5.3 Sub-Surface Hydrology

The hydrogeology of the Ciemas area has been investigated by Ir. R.M. Zahirdin Asaari (Asaari) in 2012.

Assari examined 9 DDH from the 2012 drilling program and characterised the two major lithology’s encountered: soil/unconsolidated rock and volcanic breccia as porous and semi up to impermeable respectively. Assari notes” The effect of joints and cracks in the compact rocks such as volcanic breccia might change into permeable, depends on the continuity of its joint and cracks."

The permeability classification of the two lithology's is not supported by any test work.

Mancala notes that the Ciemas project is located in the tropical monsoon climatic belt with rainfall up to 4,000mm per year. The high rain fall and relatively low relief is likely to lead to a fully charged near surface water table. The resources upon which the project is based are of a fault fissure filling style which inherently may have high transmissibility characteristics i.e. they may be a pathway for surface waters to flow to deeper depths.

Mancala recommends a detailed hydrological study be undertaken. This would entail drawn down and packer tests of selected drill holes, water level monitoring in bore holes over the entire site and an analysis of water chemistry. The hydrological consultants should be briefed to provide an estimate of potential sub-surface water inflow into the open cut workings and to recommend measures to minimise the inflow (i.e. de-watering bore holes, large scale in pit pumping etc.).

6 Mineral Resources and Data

6.1 Mineral Resources

In January 2014, PT Asia Sejati Indonesia (ASI) re-estimated the mineral resources for the Ciemas Project area using variography followed by ordinary kriging (The "ASI Report"). The resource was reported in accordance with the JORC Code (2004).

In June 2014 SRK Consulting reported an updated Mineral Resource Estimate for the Ciemas Project (SRK 2014 Report) using a data set which was essentially that same as that used by ASI. The SRK work effectively updated previous estimates made by SRK in 2013 for the project area incorporating all recent drilling data. The SRK 2014 work is reported as compliant to the 2012 JORC Code. Due to the SRK's reporting compliance and date of publication it is considered the current resource estimate for the Ciemas Project.

SRK reviewed the work of ASI in relation to resource estimates and concluded that *"there is no considerable discrepancies between the estimates* [SRK and ASI] *that may result a different conclusion from the Scoping Study"* (Table 6.1-1).

Property	Category	30 June 2014, SRK			January 2014, PT ASI		
		Resource (kt)	Au (g/t)	Au (kg)	Resource (kt)	Au (g/t)	Au (kg)
Pasir Manggu	Measured	120	7.3	870	380	7.7	2,940
	Indicated	450	7.5	3,390	200	6.8	1,330
	Inferred	270	3.8	1,030	300	3.4	1,020
Cikadu	Indicated	1,100	9.1	9,970	1,130	9.0	10,200
	Inferred	360	8.4	3,040	380	8.0	3,010
Sekolah	Indicated	710	9.2	6,520	730	8.6	6,310
	Inferred	300	8.6	2,580	320	8.6	2,730
Cibatu	Indicated	660	9.1	5,990	670	9.0	6,080
	Inferred	670	8.3	5,580	700	7.3	5,080
Total	**Measured**	**120**	**7.3**	**870**	**380**	**7.7**	**2,940**
	Indicated	**2,920**	**8.9**	**25,870**	**2,730**	**8.8**	**23,910**
	Measured + Indicated	**3,040**	**8.8**	**26,740**	**3,120**	**8.6**	**26,850**
	Inferred	**1,600**	**7.6**	**12,230**	**1,700**	**7.0**	**11,840**

Table 6.1-1 Resource estimates comparison – SRK 2014 and PT ASI, 2014. (Rounding will cause summation errors.)

Mancala notes that the Resource Estimate will be further updated in the near future incorporating the results from a recent near surface drilling program.

SRK reported the Mineral Resources for the Ciemas project, by resource area, JORC 2012 Edition classification at a 1.0g/t cut off as depicted in Table 6.1-2.

In total (Measured, Indicated and Inferred) the total Mineral Resources is:

4,640,000 tonnes at a gold grade of 8.4g/t for 1,253,000 ounces

Prospect	Classification	Tonnes (k)	Grade (g/t Au)	Au Ounces (oz)	% Total Resource oz
Pasir Manggu	Measured	120	7.3	28,000	2%
Cikadu	Measured	-	-	-	-
Sekolah	Measured	-	-	-	-
Cibatu	Measured	-	-	-	-
Total	**Measured**	**120**	**7.3**	**28,000**	**2%**
Pasir Manggu	Indicated	450	7.5	109,000	9%
Cikadu	Indicated	1,100	9.1	321,000	26%
Sekolah	Indicated	710	9.2	210,000	17%
Cibatu	Indicated	660	9.1	193,000	15%
Total	**Indicated**	**2,920**	**8.9**	**832,000**	**66%**
Pasir Manggu	Inferred	270	3.17	31,000	3%
Cikadu	Inferred	360	8.57	104,000	8%
Sekolah	Inferred	300	8.44	86,000	7%
Cibatu	Inferred	670	7.93	178,000	14%
Total	**Inferred**	**1600**	**7.6**	**393,000**	**31%**

Table 6.1-2 SRK 2014 Mineral Resource reporting of the Ciemas Project.

Mancala notes that of the global resource some 66% (by contained Au ounces) is classified as Indicated, with only 2% Measured and the remainder as Inferred. For the Pasir Manggu Resource, some 80% is Measured + Inferred. This partially reflects the density of drilling of the prospects where by Pasir Manggu is drilled on nominal 20m centres and the other prospects at 40m centres. Further infill drilling will be required to raise the classification levels of the prospects.

Mancala notes that ASI reported an apparent increase in grade with depth for all four resource areas. This may be an artefact of the interpolation process (lesser sample density) or if it is realistic it represents an important consideration with respect to underground mining. Mancala recommend a program of deep drilling below the known resources.

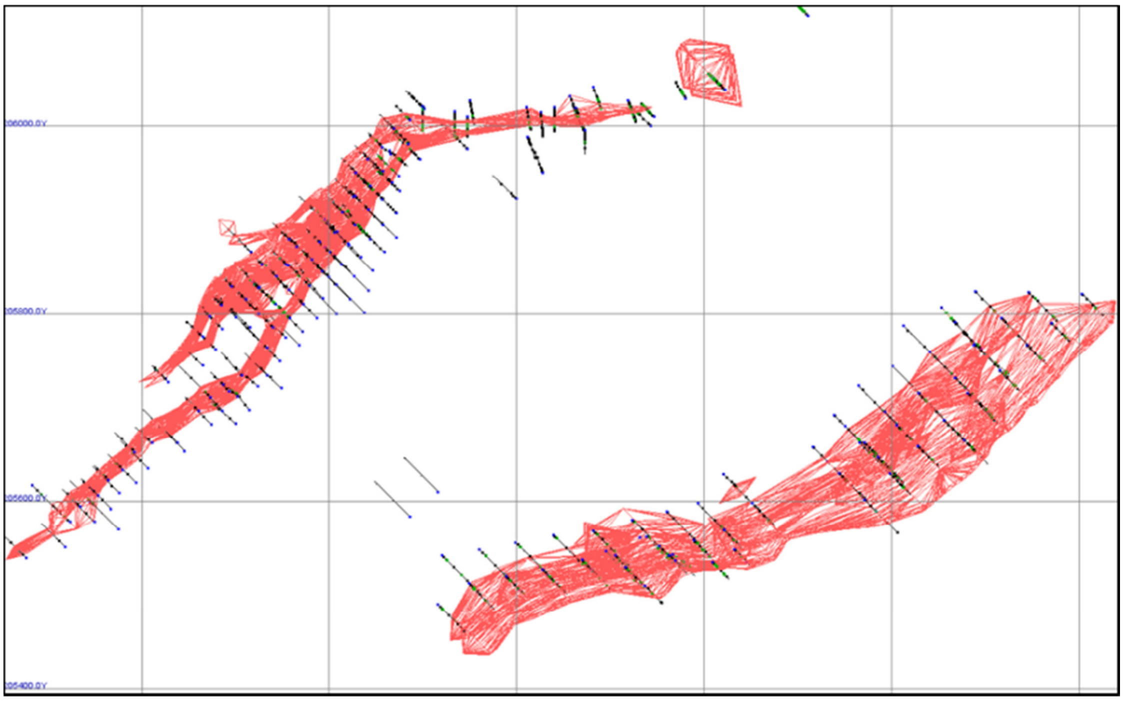


Figure 6.1-1 Plan view of Pasir Manggu (upper left) and Cikadu (lower right) resource models showing density of drilling. Grid spacing of 200m.

6.2 Bulk Density

Mancala notes that both the SRK 2014 and the ASI resource estimates have used a global specific gravity of 2.7 t/m^3 for all rock types. The block models upon which the resource is based, and upon which Mancala has based its work contains a block attribute: fresh or oxidised. ASI report that the differentiation between fresh rock and oxidised rock is based upon an interpretation from the 1990's drill sections of Parry Corporation. The transition between rocks types is depicted at between 5-30m below surface.

It is likely that the oxidised mineralisation has a specific gravity significantly less than 2.7 t/m^3. The consequence of which is that the tonnage reported in the oxide zone is probably over estimated.

Bulk density data is reported in the SKR Report as being "45 specific gravity samples were collected from the Pasir Manggu West deposit on 4th April 2012 along with 15 oxidised ore samples, 15 mixed ore samples and 15 primary ore samples ….". It is assumed the 45 samples are made up of the three lots of 15 samples. The laboratory results from this work is reported in Appendix 2 of the SRK Report and is reproduced below in Figure 6.2-1 consisting of the total data set sorted by density value.

The SRK report also notes that another batch of bulk density samples were collected for the Cikadu, Sekolah and Cibatu zone in 2012 and that average value of the fresh mineralised core is "about 2.7 t/m^3".

The SRK 2014 report does not detail any new data in relation to bulk density.

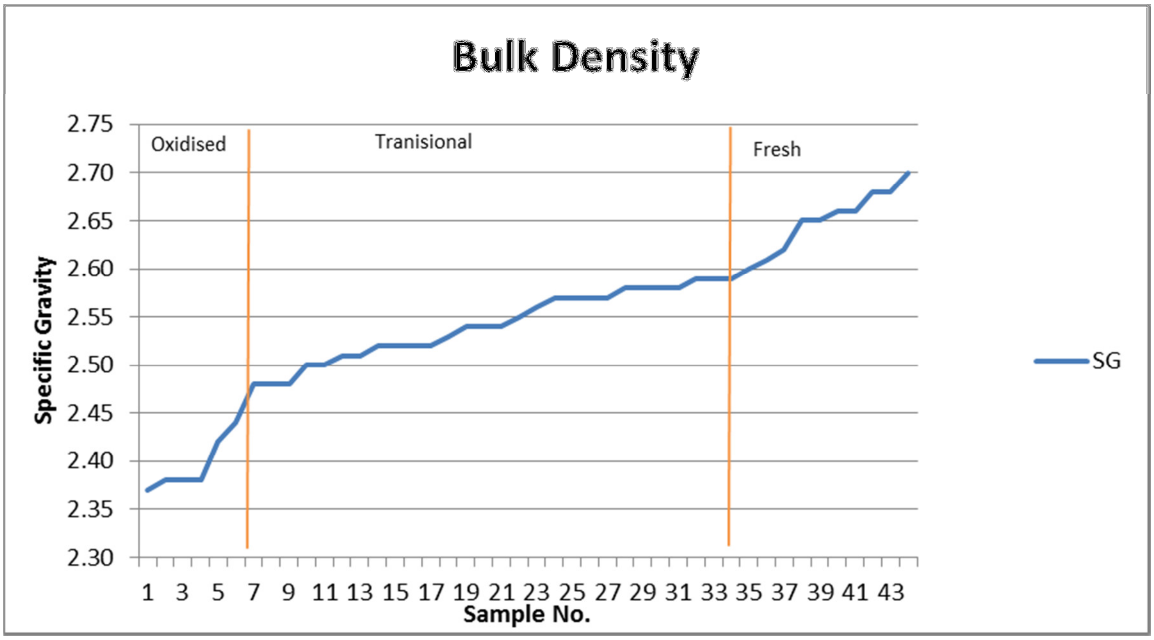


Figure 6.2-1 Bulk density results from Pasir Manggu (data from SRK Report, Appendix 2).

Within the limited specific gravity data set as plotted above, there are two breaks of slope: the first at 2.48 t/m^3 and the other at 2.59 t/m^3. The average for the three subsets is 2.42, 2.55, 2.66 t/m^3 for which it is suspected represents oxidised, transitional and fresh respectively.

For manipulation and reporting purposes Mancala has conservatively assumed all “oxidised” material within the block model has a bulk density of 2.5 t/m^3, while all fresh material has a bulk density of 2.7 t/m^3.

Mancala notes that only one sample from the Pasir Manggu samples had bulk density of 2.7 t/m^3 or above.

It is understood that a recent (2014) round of metallurgical drilling has collected core which will be subject to bulk density determination. It is recommended that the core samples be classified as fresh, transitional or oxidised and bulk density determination made accordingly.

Despite the inclusion of a lower bulk density for oxidised material, it is likely that the tonnage estimates (and thus the contained gold ounces) for near surface material is overestimated potentially between 5 - 10%. This overestimate will be exacerbated by resource depletion due to historical mining which has been reported and observed at the southern end of Pasir Manggu, Plate 6.2.1.

It is recommended that an extensive search of historical records be undertaken by a mining professional to examine all known occurrences of historical workings with potential on ground searches conducted as follow up.

Figure 6.2-2 Evidence of historical mining, southern end Pasir Manggu ore zone.

6.3 Presence of Silver in the Mineralisation

The drill hole data set provided to Mancala contains a file “DH_Collar.csv” consisting of some 606 records which describe the collar location of diamond/RC holes and costean transects converted to pseudo drill holes. Assay data associated with drill holes has been tagged by “Minzone” which represents assays occurring within the 35 individual modelled resource zones over the four prospects.

The only economic metal modelled within the resources estimate is gold. Epithermal gold deposits are frequently associated with silver occurrences, which is some instances form a significant proportion of a project’s revenue profile.

Some 1,418 Ag assay results are present within the “Minzone” data subset (i.e. occur within the modelled resource). The mean grade of these samples is 17.3g/t. Within the Pasir Manggu area (Minzone 1 to 11) being the area of greatest drilling density, some 411 Ag assays are present with a mean grade of 19.3g/t and a maximum grade 513g/t.

Although not “high grade”, the silver content of the ore would contribute to the projects revenue base, assuming the Ag is recovered in the proposed plant. At current metal pricing and assuming an 85% recovery of silver, the silver content could generate in the region of $10/t of ore. Within excess of 4.6M tonnes of ore in the resource, silver could contribute in the region of $50M to project revenue. Potential revenue will be impacted by the silver extraction method i.e. gold doré or precious metal concentrate.

Mancala recommends that all future drilling programs routinely assay for Ag, that Ag be incorporated into all future resource estimates and that process plant design consider Ag recovery along with gold.

6.4 Geological Interpretation

Mancala have been provided with 35 wireframe models (Minzone Models) representing the interpreted mineralisation's spatial extent. The interpretation was made on a cross sectional basis at 0.8g/t Au cut off. It would appear that no geological or structural constraints were used. The lack of constraints is apparently due to there only being 17 holes with lithological and structural data recorded. It would appear however, from the 1990's Parry sections that the holes were lithological and potentially structurally logged (DDH only). Locating the original logging data for all drill holes on the project is recommended.

When the interpreted ore zones are viewed both in section and plan view, there are apparently major changes in dip and strike (Figure 6.4-1 and Figure 6.4-2). Although these may be real reflections of the ore zone geometry a component of the variability could be attributed to collar and down hole survey error.

It is suspected, for instance, that hole RCH006 collar position is incorrect – its RL is too low (Figure 6.4-1).

The lack of down hole surveys (dominantly the RC holes) may impact the interpreted mineralisation's change in strike as depicted in Figure 6.5-1. On this figure, it can be seen that the mineralisation is defined by drilling on alternating sections dominated by RC and DDH. Down hole surveys for the DDH's show the holes lifted, whereas the RC holes are projected at their collar dip – where in fact they also probably lifted. The lack of survey may contribute to the interpreted sinuous ore outline.

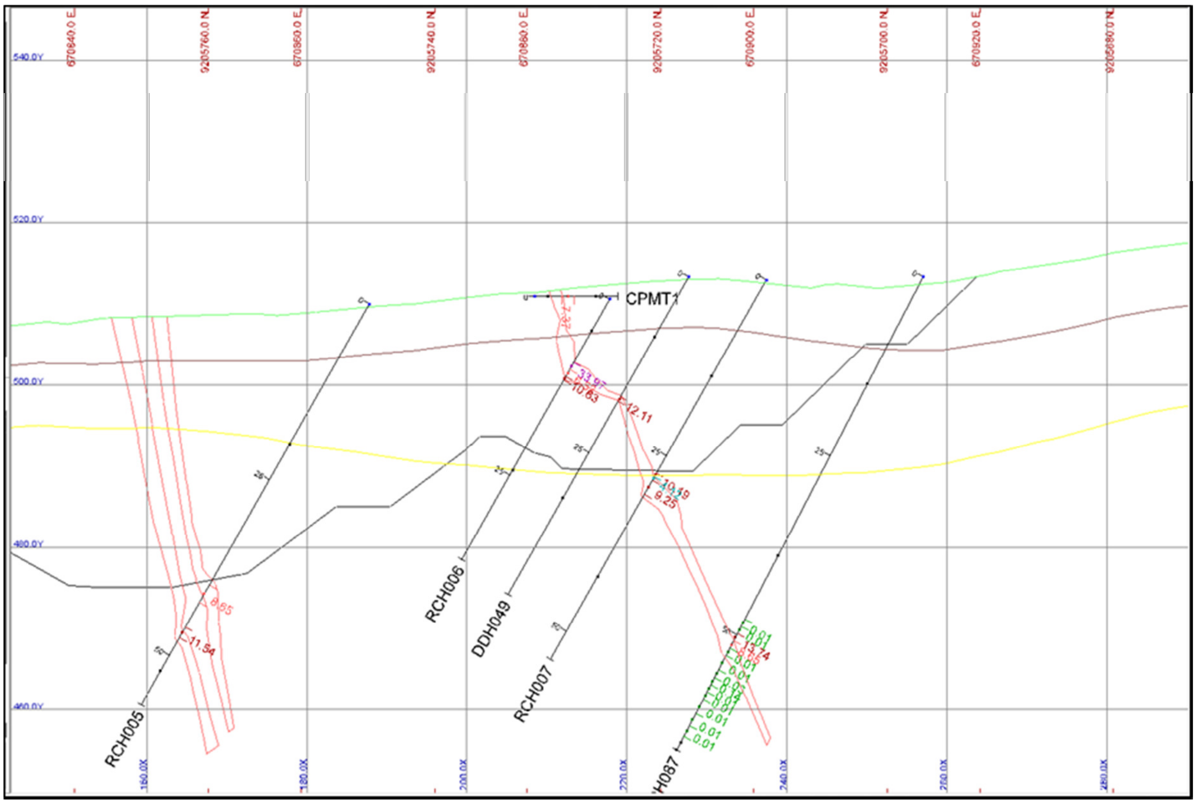


Figure 6.4-1 Section through the southern portion of Pasir Manggu ore zone showing resource outline and deflection around RCH006. Natural surface – green, soil limit – brown, oxidised limit – yellow. Proposed open cut outline – black. Grid spacing – 20m.

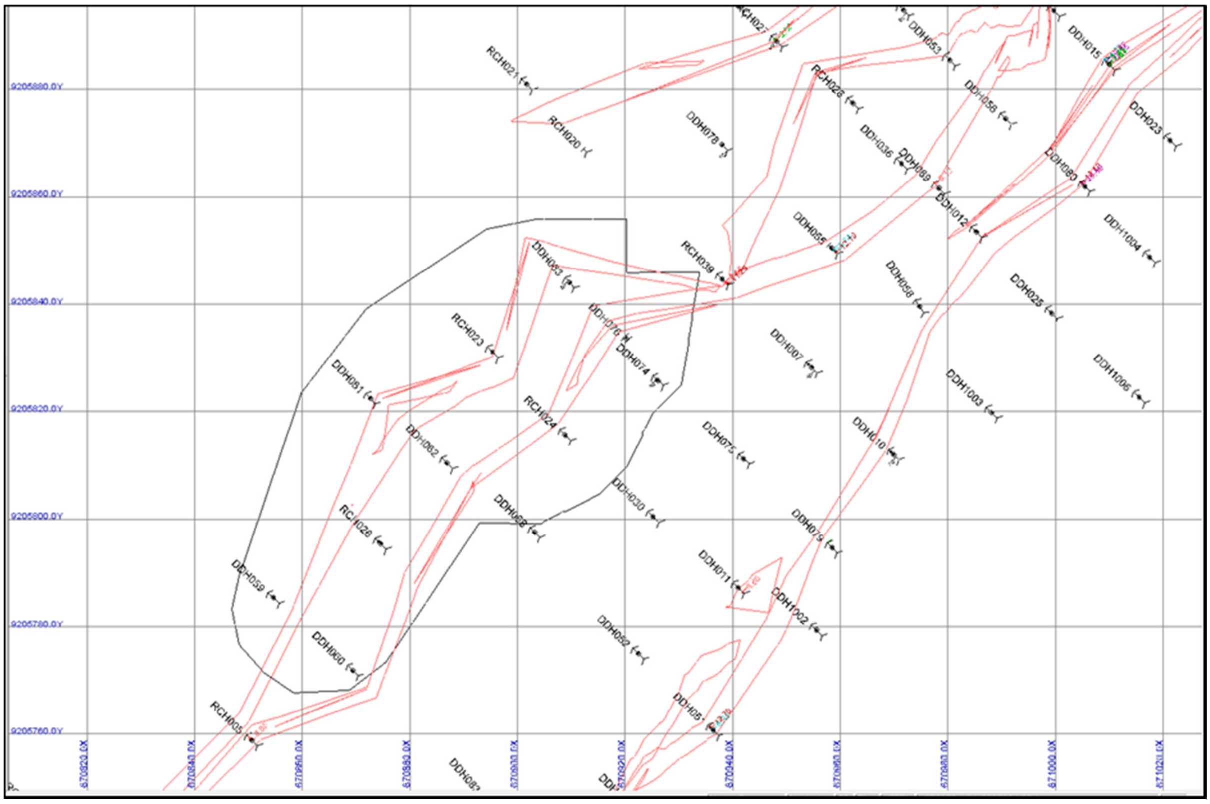


Figure 6.4-2 Plan view at 475RL of proposed Pasir Manggu open cut (black) showing ore outline (red) and DDH locations. Grid spacing 20m.

It is probable that the interpreted ore zone geometry can be refined by:

- Inclusion, where possible a lithological and structural interpretation;
- Reviewing the historical data set for additional survey and down hole survey data;
- Conducting surface re-survey of known hole locations and attempt should be made to locate holes from existing collar information;
- Conducting down hole surveys on any existing holes; and
- Examining the existing DDH down hole survey data set to determine average drill hole deviation paths and apply to the un-surveyed RC holes.

6.5 DATA VERIFICATION

Mancala has been required to import the digital drill hole information supplied by PT WWI in order become familiar with the overall geology and to perform basic statistics comparing recent to historical drilling and comparison of Au vs. As assay results.

The drill hole data set consists of historical RC and DDH holes (pre-2012), costean information compiled as pseudo drill holes and the recent (2012) DDH drilling. Also supplied to Mancala was a series of spread sheets (15) containing detailed logging (lithological and geotechnical) and some core photographs.

The SRK Report details (Table 6.5-1) the 2012 drilling program as comprising 17 holes from all resource areas with the majority of holes from Pasir Manggu (DDH1001 to DDH1006 inclusive).

The supplied digital data set contains 17 DDH holes with lithological, geotechnical and assay data, while the remainder of the holes (RC and DDH) only contain assay data. None of the 17 digital data set or the spread sheet data were from holes drilled at Pasir Manggu.

Based on the greater level of logging data, it is assumed the 17 holes in the digital data set are the 2012 drill holes, however it is uncertain why the holes differ from that listed in the SRK Report.

SRK 2012 Holes (reported in the "SRK Report")	Digital Data 2012 holes?	Spread sheet Data Holes (Logging Sheets)	Comments
DDH1001	DDH1022	DDH1022	Holes 1001-10006 in digital data
DDH1002	DDH1023	DDH1023	But no detailed logging information
DDH1003	DDH1025	DDH1025	
DDH1004	DDH1026	DDH1026	
DDH1005	DDH1036	DDH1036	
DDH1006	DDH1037	DDH1037	
DDH1021	DDH1042	DDH1042	
DDH1023	DDH1124	DDH1124	
DDH1025	DDH1131	DDH1131	
DDH1026	DDH1132	DDH1132	
DDH1031	DDH1137	DDH1138	
DDH1036	DDH1138	DDH1141	
DDH1131	DDH1138B	DDH1142	
DDH1138	DDH1141	DDH1143	
DDH1041	DDH1142	DDH1144	
DDH1042	DDH1143		
DDH1143	DDH1144		

Table 6.5-1 The 2012 drilling program and data sources.

During the importation of the drill hole data to GEMS software numerous minor errors were detected (i.e. EOH depth less than sample/lithological interval depth, survey data missing or major changes in dip/azimuth with minor depth changes). Where obvious these errors were corrected, however some more significant problems were detected.

For Hole DDH1132, data is present in the logging spread sheet and the digital data set. The sample identification numbers and samples intervals between the digital and logging sheet are inconsistent.

The digital data has the last sample at 106-107m down hole with sample number 429 (Au grade 24.3g/t) while the sample sheet has the last interval at 107-108m with sample number 435. Sample number 429 on the logging sheet refers to interval 104-105m.

For DDH 1141, the last interval in the digital data is the interval 91.6 to 92.6m with sample number 450. In the logging sheet the last interval is 91.6 to 92.6m, but the sample number is 470. Sample 450 on the logging sheet refers to an interval 74.0 to 75.0m which in the digital data has no assay result.

DDH 1144 was drilled as part of the 2012 program and appears to have been designed to duplicate a historical hole, DDH189. When viewed in section and with reference to the topographical model (Figure 6.5-1) the collar position appears some 13m too high. Clearly, this will impact on the modelled ore zone's actual spatial location.

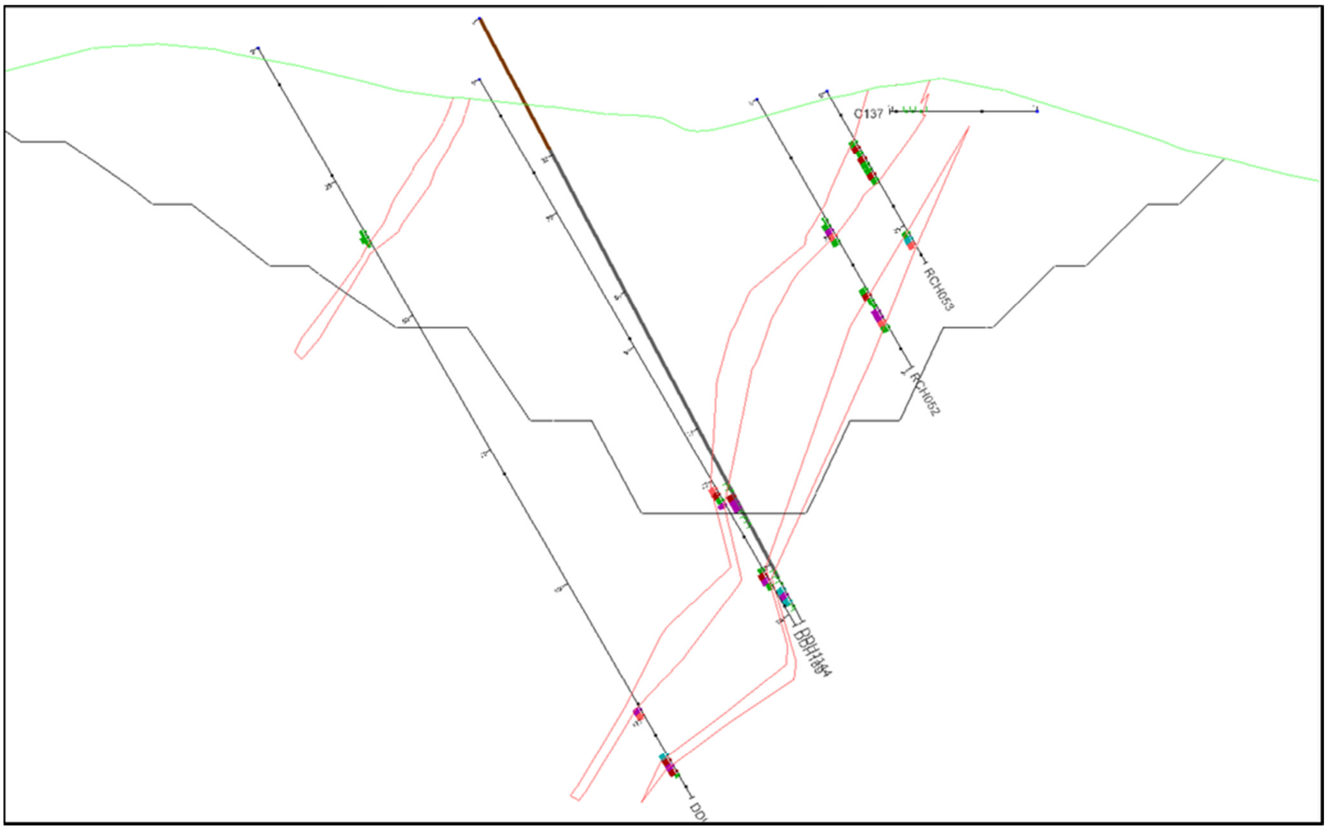


Figure 6.5-1 Drill section along DDH1144. Note collar position above the modelled topographic surface.

Mancala's verification of the digital data set has only been cursory in nature and it is recommended that an in depth examination be undertaken with reference, where possible, back to the original historical data.

6.6 Duplicate Au Data

Mancala notes that the data set constrained by the Minzone code (that being data within the resource zones) contain 67 duplicate (Au_2) Au assays. Duplicates are routinely reported by assay laboratories by analysing material from the same sample to gauge the accuracy of the original result. A significant variation between the original result and the duplicate would suggest either a high nugget effect and/or inappropriate sample preparation procedures producing an inhomogeneous sub-sample.

Although the data set is small (67 duplicates) a high degree of correlation is present at the typical resource grade samples (2.5-10g/t Au) with lesser correlation at higher grades (Figure 6.6-1).

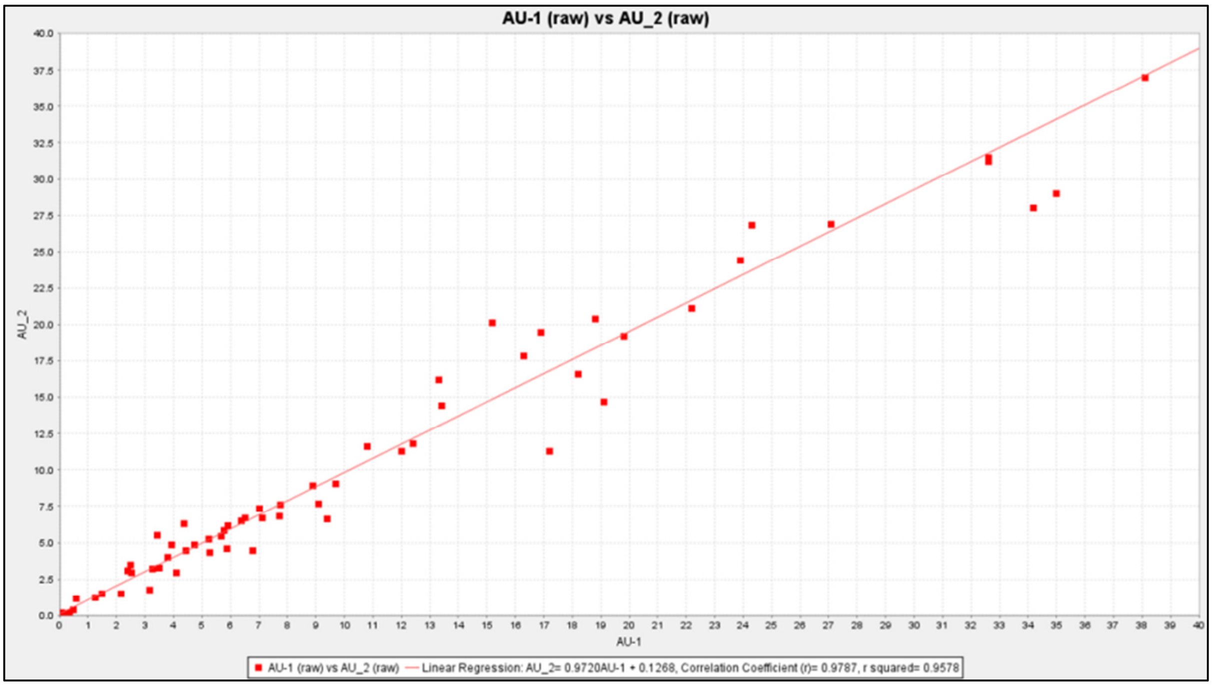


Figure 6.6-1 Correlation matrix Au_1 and Au_2 from Minzone 1 to 35.

6.7 HISTORICAL VS. 2012 DRILLING PROGRAM ASSAY RESULTS

The 2012 drilling program was instigated and supervised by SRK, using an established QA/QC program. The historical data however is unverified. Although there is no reason to suppose any bias may exist in the historical data, a comparison of it to the 2012 data is warranted.

Figure 6.7-1 displays a cumulative frequency plot of pre 2012 Au assays within Minzone 1-35 and the 2012 drilling program assays results within Minzone 1-35. Close correlation is apparent, particularly at grades similar to the reported total resource grade (2012 drilling 115 samples and pre 2012 drilling 975 samples).

From the available data there appear to be little or no bias between pre 2012 and 2012 drilling programs in terms of Au analysis.

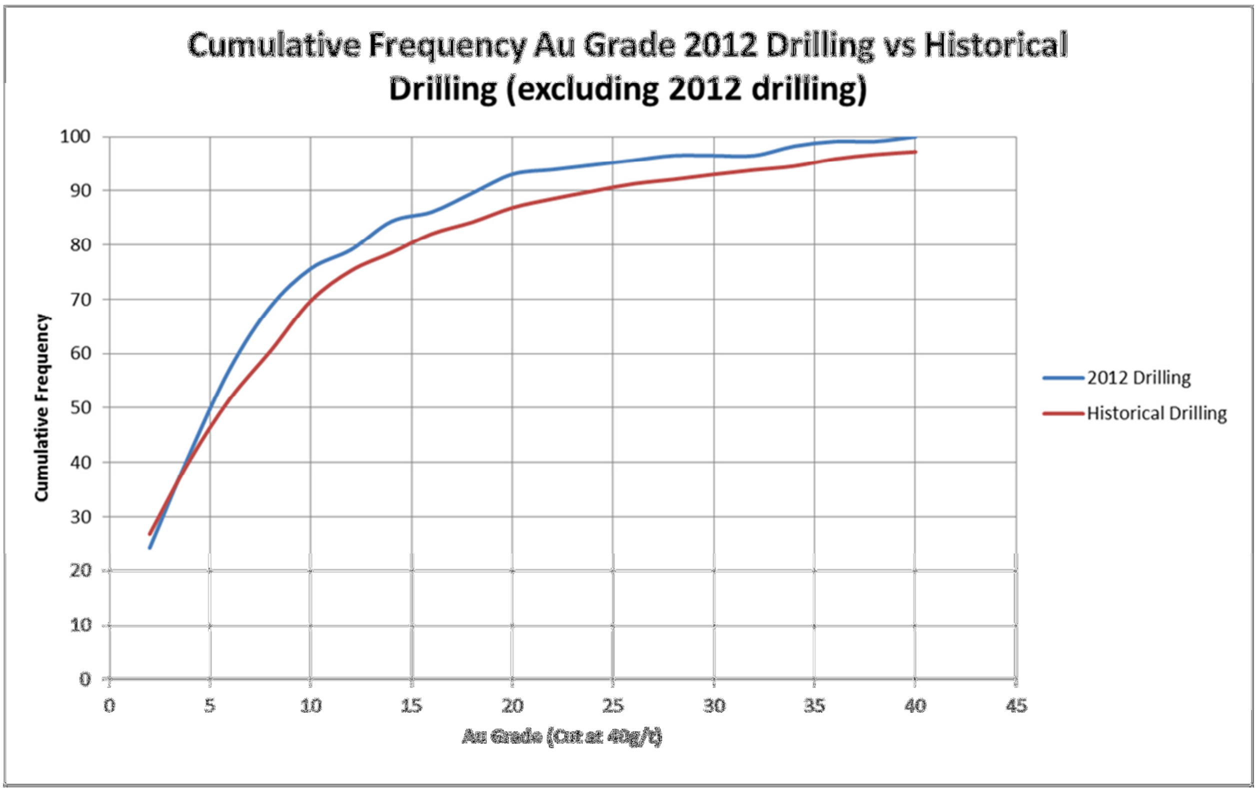


Figure 6.7-1 Pre 2012 Au assay results vs. the 2012 drilling program.

6.8 Au vs. As Analytical Results

The digital data set when constrained by Minzone 1 to 35 contains 1,105 samples that have had both Au and arsenic (As) analysis preformed. A positive relationship between As and Au (i.e. high As infers high Au grade) can be very useful for grade control purposes when hand held XRF tools are used. The tool can immediately analyse for As and an Au grade can be inferred.

To test the relationship between the two elements a simple scatter plot has been produced (Figure 6.8-1). Unfortunately there is no discernable relationship in the data. High As grades are associated with both high and low Au grades.

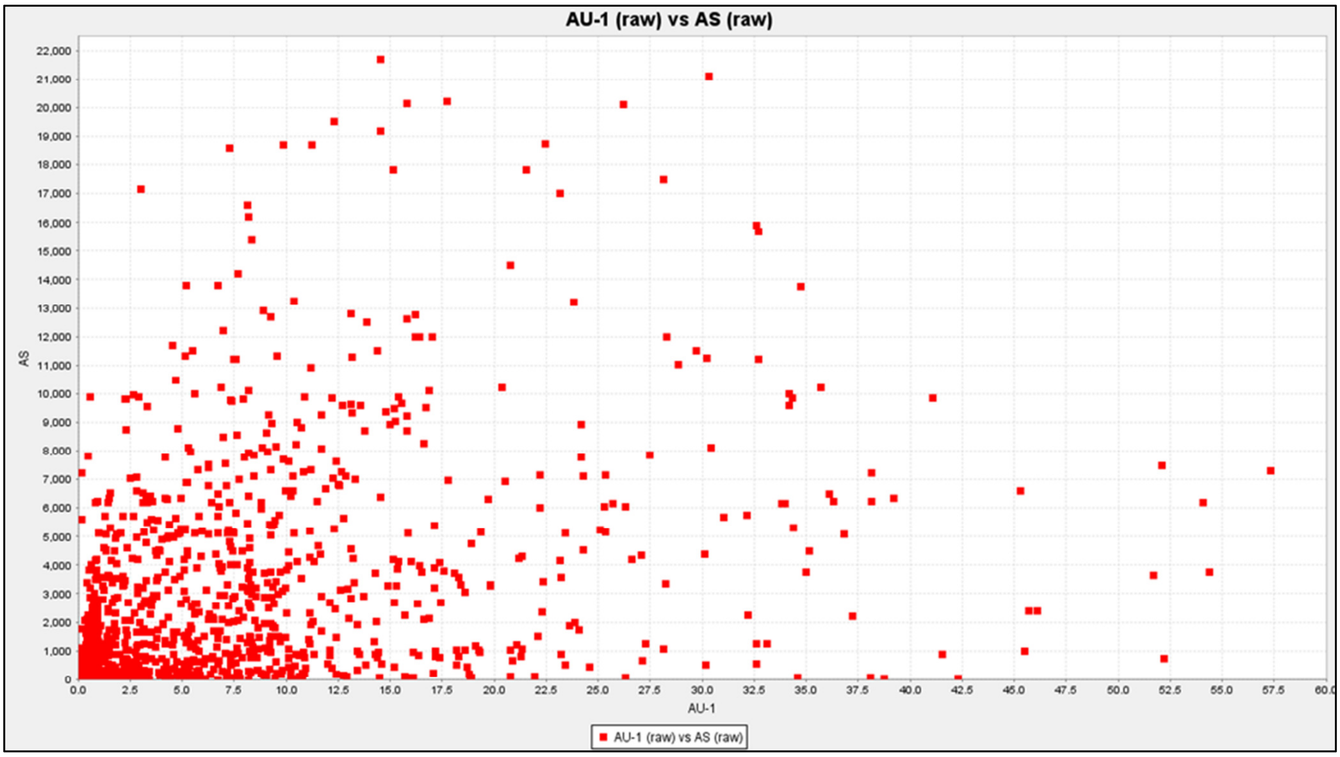


Figure 6.8-1 Scatter plot Au_1 and As Minzone 1-35.

7 MINE PLANNING

The Ciemas project consists of four Mineral Resource areas, namely Pasir Manggu (PSM), Cikadu (CKD), Sekolah (SEK) and Cibatu (CBT) and a significant number of partially defined exploration prospects. The resource zones are open at depth and in some instances along strike.

The Pasir Manggu resource has been modelled to comprise 11 separate sub-zones termed PSM01 to PSM 11. Cikadu, Sekolah and Cibatu are similarly sub-divided; CKD12 - CKD19, SEK20 – SEK29 – SEK30 and CBT31 – CBT35.

The resource areas are located in gently rolling topography, generally sloping to the southeast with a maximum elevation differential over the site of 70m. The area is incised by meandering water courses whose flow is generally south eastward and in some instances transects the resource zones Figure Ex.1.1-1.

The strike extent of the defined resources is some 2.8km and the resource footprint is some 117 hectares.

The deposits are hosed by persistent and apparently continuous (vertically and horizontally) fault structures oriented either NW/SE or NE/SW. Mineralisation with the fault structures averages 4.0m true width, is sub vertical to some 60 degrees from horizontal. The maximum interpreted depth below surface is some 160m.

7.1 Historical Mine Planning

In 2012, the Yantai Design Engineering Co. Ltd. (Yantai) conducted a technical study to mine the Ciemas gold resources by underground methods. Three mines centred upon the PSM, CKD and SEK/CBT resources were planned, producing 400, 500 and 600 tonnes/day respectively.

The PSM mine was to be accessed via an adit, a main shaft, and auxiliary shaft and an inclined shaft. The mine was to be developed at 40m level intervals. Short hole shrink stoping was chosen as the primary mining method. Waste and ore haulage were contemplated for the incline and main shafts with the adit and auxiliary shaft used for man and materials transport.

The other two mines were to be accessed and developed in a similar manner to PSM, with all development and stoping conducted by hand held means. No fill was envisaged in the mine plan with long term stability being afforded by rib pillars (60m spacing) and crown pillars. A recovery rate of 85% and dilution of 20% was envisaged by Yantai.

SRK Report and Yantai defined an Ore Reserve reported to be in accordance with the JORC 2004 Code, reproduced in Table 7.1-1 below.

Property	Category	Reserve (kt)	Au (g/t)	Au (kg)	Au ('000 oz)
Pasir Manggu	Proved	100	5.9	6010	20
	Probable	460	6.6	3,000	100
	Proved + Probable	600	6.5	3,610	120
Cikadu	Probable	840	7.3	6,190	200
Sekolah	Probable	430	7.9	3,400	110
Cibatu	Probable	600	6.8	4,130	133
Total	Proved	100	5.9	610	20
	Probable	2,340	7.2	16,730	540
	Proved + Probable	2,440	7.1	17,330	560

Table 7.1-1 Ore Reserve estimate of SRK Report (2013) for hand held underground mining methods. Table modified from SRK by rounding to reflect perceived accuracy, summation errors will result.

SRK also report on the anticipated capital and operating costs and model the project financial return, in terms of NPV at a 10% discount as being $280M. The estimated capital costs and unit operating costs are reproduced below in Table 7.1-2 and Table 7.1-3. .

Item	Capital (USD '000)	% of Total
Mining	32,918	35
Processing	18,000	19
Tailings Storage Facility	9,000	10
Water Supply	700	1
Power Supply	1,575	2
Infrastructure	14,000	15
General Lay Out	3,000	3
Others	13,557	15
Total	**92,750**	**100**

Table 7.1-2 Capital cost estimate of SRK Report (2013).

Item	Value (USD/t Ore)	% of Total
Mining Cost	22.60	3.42
Processing Cost	20.78	31.5
Administration, Sales and other	22.62	34.3
Total	66.00	100

Table 7.1-3 Unit Production Cost of SRK Report (2013).

The possibility of open cut mining appears not to have been considered by Yantai.

7.2 Mining Methodology and Approach

In selecting the basic mining method, the constraints and opportunities of the ore-body must be considered and in general the best method will be that which optimises both mining performance and cost. Intrinsically, near surface or outcropping ore bodies are best exploited by open cut means. Open cut operations are attractive in these situations due to low mining cost; flexibility in ore scheduling; high recovery of the resource; ability to accurately define the resource as the excavation advances and common operator skill sets.

The principal parameters to be considered in selecting a mining method are:

- Safety;
- Mining stability (ground conditions);
- Ore recovery and dilution;
- Mining cost;
- Impact on milling costs and performance;
- Flexibility of production (grade and production rate);
- Productivity; and
- Simplicity, skill requirements.

For a deposit like Ciemas, where a high grade resources outcrop, near surface ground conditions are poor and ore continuity is not certain, open cut mining would appear a sensible approach. The maximum depth that an open cut operation should extend to can determined by pit optimisation software. This produces a series of nested pit shells, from which the user can select the one which optimises NPV.

Once an open cut has reached its most economic designed depth, underground mining must thereafter be considered for the remaining resource extraction.

Open cut mining by necessity results in a far larger foot print of modified land than does underground mining. Open cut operations require land to be disturbed above and adjacent to the ore body and require significant land area for the construction of waste dumps.

On closure, an open cut operation will have waste dumps which require rehabilitation. The abandoned open cuts commonly fill with water, forming artificial lakes.

Thus in instances where open cut operations are restricted due to environmental, social, political or land use issues and underground operation may be preferred.

Mancala have been advised by PT WWI that there are no surface restrictions for open cut operations at Ciemas. Mancala notes however, and as reported in the SRK Report, only some 28 hectares of land use rights have been negotiated by PT WWI vs. the mineral resources footprint of 117 hectares.

7.3 Geotechnical Considerations

Knowledge of the strength, structure and stability of the rock mass through which either open cut or underground operations are developed is critical for technical design purposes and for accurate cost estimation.

The rock mass in the Ciemas area was reported upon by Ir. Alwin Darmawan in 2012. In this report, four drill holes (DDH 1003, 1021, 1031, and 1041) are described lithologically along with a description of the partially developed incline shaft in the Pasir Manggu area. Samples were taken from the holes and shaft location for laboratory analysis of point load and unconfined compressive strength.

Laboratory results indicate the rocks sampled are medium strong to strong with UCS's from 31.3 to 62.8 MPa.

The samples tested by Ir. Alwin Darmawan are all from relatively deep locations. The shallowest sample is from DDH1041 at 45m below surface and the average depth of the remaining drill hole samples is 112m below surface.

The detailed geotechnical information provided in the logging sheets from the 2012 program appears not to have been sighted by Ir. Alwin Darmawan.

In March 2014, Mancala commissioned Ground Control Engineering (GCE) to:

- Determine the major structural elements, domain lithologically and/or by structure type and present their analysis as stereo nets;
- Describe structures and comment on their likely stability with reference to the proposed open cuts;
- Provide comments on dig ability vs. depth based on the available information; and
- Recommend procedures/practices for the collection of additional geotechnical data (additional 30 holes planned).

Mancala supplied to GCE the geotechnical and geological data collected from 15 exploration drill holes (spread sheet data), the Geology and Geomechanic Report of Ciemas Gold Project (Alwin) and dtm's of conceptual pit shells and mineralised lenses.

Mancala notes that the geotechnical information is limited to 15 drill holes, with no holes intersecting the Pasir Manggu resource zone. The data was provided on the logging sheets (Table 7.3-1). The GCE report is included as Appendix D.

PT. ASIA SEJATI INDUSTRI
GEOLOGICAL & GEOTECHNICAL DRILL LOG

PROJECT : Ciemas
PROSPECT : Cibatu
HOLE_ID : DDH - 1042

NORTHING : 9206629.000
EASTING : 673223.141
RL : 535.400 m
COLLAR AZIMUTH : N 134.7° E
COLLAR DIP : -71.9

TOTAL DEPTH : 96.7 meter
RIG TYPE : Jecro
CONTRACTOR : PT. Sugih Jaya Tata Lestari
DRILL START DATE : 1-Oct-12
DRILL FINISH DATE : 14-Oct-12

LOGGED BY : Airlangga & Candra
DATE :
CHECKED BY :
DATE :
PAGE : OF

From (m)	To (m)	GEOLOGY DATA: Lithology	Colour	Grainsize	Weathering	Strength	Vein Qual	Stockwork	Alteration 1	Alteration 2	Alt. Intensity	Minerals: Mineral 1 Min	Mineral 1 Style	Mineral 1 Int	Mineral 2 Min	Mineral 2 Style	Mineral 2 Int	Mineral 3 Min	Mineral 3 Style	Mineral 3 Int
0.00	0.70	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
0.70	2.20	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
2.20	3.70	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
3.70	5.20	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
5.20	6.70	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
6.70	8.20	Sol	Dark Brown	Clay	CW	R0	-	-	-	-		-	-	-	-	-	-	-	-	-
8.20	9.70	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
9.70	11.20	Sol	Red Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
11.20	12.70	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
12.70	14.20	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
14.20	15.70	Sol	Dark Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
15.70	17.20	Sol	Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
17.20	18.70	Sol	Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
18.70	20.20	Sol	Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
20.20	21.70	Sol	Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
21.70	23.20	Sol	Light Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
23.20	24.70	Sol	Light Brown	Clay	CW	R0	-	-	-	-	-	-	-	-	-	-	-	-	-	-
24.70	26.20	Sol	Brown	Clay	CW	Ro	-	-	-	-	-	-	-	-	-	-	-	-	-	-
26.20	27.70	Sol	Brown	Clay	PW	R1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
27.70	29.20	Sol	Brown	Clay-Fine grain	PW	R2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
29.20	30.70	VBR	Greenish Grey	Lapili-Block	PW	R4	-	-	-	-	-	-	-	-	-	-	-	-	-	-
30.70	32.20	VBR	Grey	Lapili-Block	FO	R3	-	Au?	SI		ST	-	-	-	-	-	-	-	-	-
32.20	33.70	VBR	Grey	Lapili-Block	FO	R3	-	Au?	SI		ST	-	-	-	-	-	-	-	-	-
33.70	35.20	VBR	Greenish Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
35.20	36.70	VBR	Greenish Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
36.70	38.20	VBR	Greenish Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
38.20	39.70	VBR	Greenish Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
39.70	41.20	VBR	Greenish Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
41.20	42.70	VBR	Greenish Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
42.70	44.20	VBR	Grey	Lapili-Block	SW	R3	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
44.20	45.70	VBR	Grey	Lapili-Block	PW	R3	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
45.70	47.20	VBR	Grey	Lapili-Block	PW	R3	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
47.20	48.70	VBR	Grey	Lapili-Block	SW	R4	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
48.70	50.20	VBR	Grey	Lapili-Block	SW	R5	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
50.20	51.70	VBR	Grey	Lapili-Block	SW	R6	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
51.70	53.20	VBR	Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
53.20	54.70	VBR	Grey	Lapili-Block	SW	R3	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
54.70	56.20	VBR	Grey	Lapili-Block	SW	R3	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
56.20	57.70	VBR	Grey	Lapili-Block	SW	R3	CAL	QC	PR		ST	PY	Disseminated	<2%	-	-	-	-	-	-
57.70	59.20	VBR	Grey	Lapili-Block	SW	R3	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
59.20	60.70	VBR	Grey	Lapili-Block	SW	R3	CAL	QC	PR		ST	-	-	-	-	-	-	-	-	-
60.70	62.20	VBR	Grey	Lapili-Block	UW	R4			PR		WE	-	-	-	-	-	-	-	-	-
62.20	63.70	VBR	Grey	Lapili-Block	UW	R4			PR		WE	-	-	-	-	-	-	-	-	-
63.70	65.20	VBR	Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
65.20	66.70	VBR	Grey	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
66.70	68.20	VBR	Dark Grey	Lapili-Block								-	-	-	-	-	-	-	-	-
68.20	69.70	VBR	Dark Grey	Lapili-Block	UW	R6	CAR		PR-CHL		WE-ST	-	-	-	-	-	-	-	-	-
												-	-	-	-	-	-	-	-	-
69.70	71.20	VBR	Dark Grey	Lapili-Block	UW	R6		-	PR-CHL		WE-ST	-	-	-	-	-	-	-	-	-
												-	-	-	-	-	-	-	-	-
												-	-	-	-	-	-	-	-	-
71.20	72.70	VBR	Grey	Lapili-Block	SW	R2		CB	PR		VS	-	-	-	-	-	-	-	-	-
72.70	74.20	VBR	Grey	Lapili-Block	SW	R2		CB	PR		VS	-	-	-	-	-	-	-	-	-
74.20	75.70	VBR	Grey	Lapili-Block	SW	R2		CB	PR		VS	-	-	-	-	-	-	-	-	-
75.70	77.20	VBR	Grey	Lapili-Block	SW	R2		CB	PR		VS	-	-	-	-	-	-	-	-	-
77.20	78.70	VBR	Grey	Lapili-Block	UW	R4	-	-	PR		WE	-	-	-	-	-	-	-	-	-
78.70	80.20	VBR	Grey, Black	Lapili-Block	UW	R4	-	-	PR	-	WE	-	-	-	-	-	-	-	-	-
80.20	81.70	VBR	Grey	Lapili-Block	VW	R3-R4			SIL		ST	PY	Spotted	2-5%	-	-	-	-	-	-
81.70	83.20	VBR	Grey	Lapili-Block	PW	R2-R3			AR		WE	Py	Spotted	7-10%	-	-	-	-	-	-

From (m)	To (m)	Structure Zone: From	TO	Structure	Angle	GEOTECHNICAL DATA: Core Interegity: Recovery Meter	Recovery (%)	RQD Length >10 cm	RQD (%)	Core_Type	Structure Event: Depth (m)	Structure Type	Quantity	Alpha	Beta	Roughness	Infill Min.1	Infill Min.2	Infill Width (mm)	Orientation	Ori Quality	SAMPLE NO.	INTERVAL	ROCK TYPE	GRAPHIC LOG	REMARKS
0.00	0.70	-	-	-	-	0.3	42.86	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
0.70	2.20	-	-	-	-	0.7	46.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
2.20	3.70	-	-	-	-	1	66.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
3.70	5.20	-	-	-	-	1.5	100.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
5.20	6.70	-	-	-	-	1.5	100.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
6.70	8.20	-	-	-	-	0.5	33.33	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
8.20	9.70	-	-	-	-	0.9	60.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
9.70	11.20	-	-	-	-	1	66.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Reddish Brown
11.20	12.70	-	-	-	-	1.4	93.33	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
12.70	14.20	-	-	-	-	0.4	26.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown, mineral hematite
14.20	15.70	-	-	-	-	1.5	100.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
15.70	17.20	-	-	-	-	1.5	100.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Brown,Oxidation
17.20	18.70	-	-	-	-	1.5	100.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
18.70	20.20	-	-	-	-	0.4	26.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
20.20	21.70	-	-	-	-	0.4	26.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, Dark Brown
21.70	23.20	-	-	-	-	1.5	100.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, light brown, fine grain , clay - sand, complete oxidation .
23.20	24.70	-	-	-	-	1.15	76.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, brown fine grain
24.70	26.20	-	-	-	-	1.5	100.00	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, light brown, fine grain,
26.20	27.70	-	-	-	-	1.3	86.67	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil , light brown , coarse grain, fragment andesitic , moderat oxidation
27.70	29.20	-	-	-	-	1.49	99.33	0	0.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Soil, light brown, coarse grain (sand - gravel) , fragment andeitic (1 - 5 cm), moderate weathering
29.20	30.70	-	-	-	-	1.3	86.67	0.27	18.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic breccia, gray greenish, coarse grain , sand - gravel, fragment andesitic min plagioklas , weak alteration, propilitic , mineral chlorie, vein let calcite, low- moderate oxidation
30.70	32.20	-	-	-	-	1.5	100.00	0.69	46.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Gold (Au), Calcite, Grey, Massive, Subhedral, Breccia
32.20	33.70	-	-	-	-	1.5	100.00	0.85	56.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Gold (Au), Calcite, Grey, Massive, Subhedral, Breccia
33.70	35.20	-	-	-	-	1.5	100.00	1.04	69.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic Breccia, grey, greenish, coarse grain (sand - gravel), poorly sorted, fragment andesitic, kompak, mineral quartz, calcite , chorite, etc. weak alteration propilitic.
35.20	36.70	-	-	-	-	1.5	100.00	1.34	89.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic Breccia, grey, greenish, coarse grain (sand - gravel), poorly sorted, fragment andesitic, kompak, mineral quartz, calcite , chorite, etc. weak alteration propilitic.
36.70	38.20	-	-	-	-	1.5	100.00	1.48	98.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic Breccia, grey, greenish, coarse grain (sand - gravel), poorly sorted, fragment andesitic, kompak, mineral quartz, calcite , chorite, etc. weak alteration propilitic.
38.20	39.70	-	-	-	-	1.5	100.00	1.12	74.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic Breccia, grey, greenish, coarse grain (sand - gravel), poorly sorted, fragment andesitic, kompak, mineral quartz, calcite , chorite, etc. weak alteration propilitic.
39.70	41.20	-	-	-	-	1.5	100.00	1.34	89.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic Breccia, grey, greenish, coarse grain (sand - gravel), poorly sorted, fragment andesitic, kompak, mineral quartz, calcite , chorite, etc. weak alteration propilitic.
41.20	42.70	-	-	-	-	1.5	100.00	1.43	95.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic Breccia, grey, greenish, coarse grain (sand - gravel), poorly sorted, fragment andesitic, kompak, mineral quartz, calcite , chorite, etc. weak alteration propilitic.
42.70	44.20	-	-	-	-	1.5	100.00	1.32	88.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
44.20	45.70	-	-	-	-	1.5	100.00	1.32	88.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
45.70	47.20	-	-	-	-	1.5	100.00	1.29	86.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
47.20	48.70	-	-	-	-	1.5	100.00	0.9	60.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
48.70	50.20	-	-	-	-	1.5	100.00	1.28	85.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
50.20	51.70	-	-	-	-	1.5	100.00	1.26	84.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
51.70	53.20	-	-	-	-	1.5	100.00	1.24	82.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Volcanic Breccia, grey, greenish, coarse grain (sand - gravel), poorly sorted, fragment andesitic, kompak, mineral quartz, calcite , chorite, etc. weak alteration propilitic.
53.20	54.70	-	-	-	-	1.5	100.00	1.01	67.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
54.70	56.20	-	-	-	-	1.5	100.00	0.94	62.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
56.20	57.70	-	-	-	-	1.5	100.00	0.6	40.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
57.70	59.20	-	-	-	-	1.5	100.00	1.18	78.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
59.20	60.70	-	-	-	-	1.5	100.00	0.34	22.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Calcite, Quartz, Chlorite,Massive, Subhedral, Volcanic Breccia
60.70	62.20	-	-	-	-	1.5	100.00	0.85	56.67	HQ	-	-	-	-	-	-	-	-	-	-	-					
62.20	63.70	-	-	-	-	1.5	100.00	0.74	49.33	HQ	-	-	-	-	-	-	-	-	-	-	-					
63.70	65.20	-	-	-	-	1.5	100.00	1.01	67.33	HQ	-	-	-	-	-	-	-	-	-	-	-					
65.20	66.70	-	-	-	-	1.5	100.00	1.3	86.67	HQ	-	-	-	-	-	-	-	-	-	-	-					
66.70	68.20	-	-	-	-	1.5	100.00	0.75	50.00	HQ	-	-	-	-	-	-	-	-	-	-	-					
68.20	69.70	68.47	68.53	VL	45	1.5	100.00	1.13	75.33	HQ	68.47	VL	1	40	54	-	CAR	-	-	1	-					Not vein alteration to mineralisation, have a fresh rock of volcanic breccia
		69.41	69.48	VL	63						69.41	VL	1	340	65	-	CAR	-	-	3	-					
69.70	71.20	70.05	70.08	VL	50	1.5	100.00	1.34	89.33	HQ	70.05	VL	1	313	76	-	CAR	-	-	1	-					Not vein alteration,mineralisation, have a fresh rock of volcanic breccia; intens of calcite veinlets
		70.28	70.35	VL	46						70.28	VL	1	318	62	-	CAR	-	-	1	-					
		70.95	71.00	VL	40						70.95	VL	1	2	30	-	CAR	-	-	2	-					
71.20	72.70	-	-	-	-	1.5	100.00	1.43	95.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Calcite, Chlorite,Clustiform, Subhedral, Fine Graned, Volcanic Breccia
72.70	74.20	-	-	-	-	1.5	100.00	1.05	70.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Calcite, Chlorite,Clustiform, Subhedral, Fine Graned, Volcanic Breccia
74.20	75.70	-	-	-	-	1.5	100.00	1.28	85.33	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Calcite, Chlorite,Clustiform, Subhedral, Fine Graned, Volcanic Breccia
75.70	77.20	-	-	-	-	1.5	100.00	0.96	64.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Calcite, Chlorite,Clustiform, Subhedral, Fine Graned, Volcanic Breccia
77.20	78.70	-	-	-	-	1.5	100.00	0.55	36.67	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Calcite, Chlorite,Clustiform, Subhedral, Fine Graned, Volcanic Breccia
78.70	80.20	-	-	-	-	1.5	100.00	0.96	64.00	HQ	-	-	-	-	-	-	-	-	-	-	-					Quartz, Calcite, Chlorite,Clustiform, Subhedral, Fine Graned, Volcanic Breccia
80.20	81.70	81.70	81.79	VL	80	1.5	100.00	0.45	30.00	HQ	81.70	VL	1	245	80	-	-	-	2		-					
81.70	83.20	82.18	82.48	VL	48	1.5	100.00	0.13	8.67	HQ	82.18	VL	1	51	45	-	-	-	2		-					Rich mineralisation

Table 7.3-1 Geotechnical logging sheet example.

GCE separated the structural data into the major lithology's (VBR, TUFF and LPS) and analysed the data, following Terzaghi correction via stereonets. The stereonets were used to identify the major structural elements and define structural sets as outlined in Table 7.3-2.

Rock Unit	Structure Type	Major Structural Set Orientations			# of Measurements in Data Set	Comments
		Set 1	Set 2	Set 3		
VBR	SHR	71/199	71/350	28/306	243	
	VL/VN	62/214	63/358	28/257	360	
	FV	Present	83/003		12	
TUFF	SHR	Present	Present		15	Insufficient data to define specific set orientation(s)
	VL/VN	Present	Present		9	
LPS	SHR	Present	Present		8	
	VL/VN	Present		Present	8	

Table 7.3-2 Structural data sets after GCE.

GCE's comments as to the potential impact of the structural sets on the proposed open cuts are reproduced below:

7.3.1 Structural Set 1:

Structural Set 1 is a dominant structural orientation in the data set and consists of steep south to south-south west dipping structures. Set 1 is evident in all structure types across the three major rock types in the data set. This set has the potential to contribute to topple style failure on the southern walls of the proposed open cuts. The likelihood and potential for topple failure requires further detailed assessment and depends on the open cut wall angles, the spacing and persistence of structural discontinuities (block size) and the shear strength properties of the Set 1 defects.

Set 1 is unlikely to be associated with discrete planar wedge failure on this structural set alone due to it being relatively steeply dipping and unlikely to daylight in the open cut walls. It may however, provide a bounding release structure for potential wedge failure on the eastern open cut walls, associated with Structural Set 3.

7.3.2 Structural Set 2:

Structural Set 2 is a similarly dominant structural orientation in the data set and consists of steep north to north-north west dipping structures. Set 2 is evident in all structure types across the three major rock types in the data set. This set has the potential to contribute to topple style failure on the northern walls of the proposed open cuts. As with Set 1, the likelihood and potential for topple failure requires further detailed assessment and depends on the open cut wall angles, the spacing and persistence of structural discontinuities (block size) and the shear strength properties of the Set 2 defects.

As with Set 1, Set 2 is unlikely to be associated with discrete planar wedge failure on this structural set alone due to it being relatively steeply dipping and unlikely to daylight in the open cut walls. It may however, provide a bounding release structure for potential wedge failure on the eastern open cut walls, associated with Structural Set 3.

7.3.3 Structural Set 3:

Structural Set 3 consists of relatively flat, westerly dipping structures. Set 3 is a less dominant structural orientation and is only evident in the SHR and VL/VN structures in the VBR rock unit and in the VL/VN structures in the LPS rock unit.

Set 3 has the potential to contribute to planar and wedge style failure on the eastern walls of the proposed open cuts. The overall potential for wedge failure is limited by the flat dip of the Set 3 structures which ranges from approximately 15° to 35°. If wedge failure is to occur, the friction angle of the defect(s) must be less than the dip of the sliding plane. GCE understand that no test data regarding defect shear strength properties is available at this stage however, friction angles of less than 30° are typically associated with the presence of soft infill materials on the defect surfaces.

The potential for wedge failure is somewhat increased by the presence of structural sets 1 and 2 which may form bounding release planes if they are found to intersect Set 3 structure in the open cut walls.

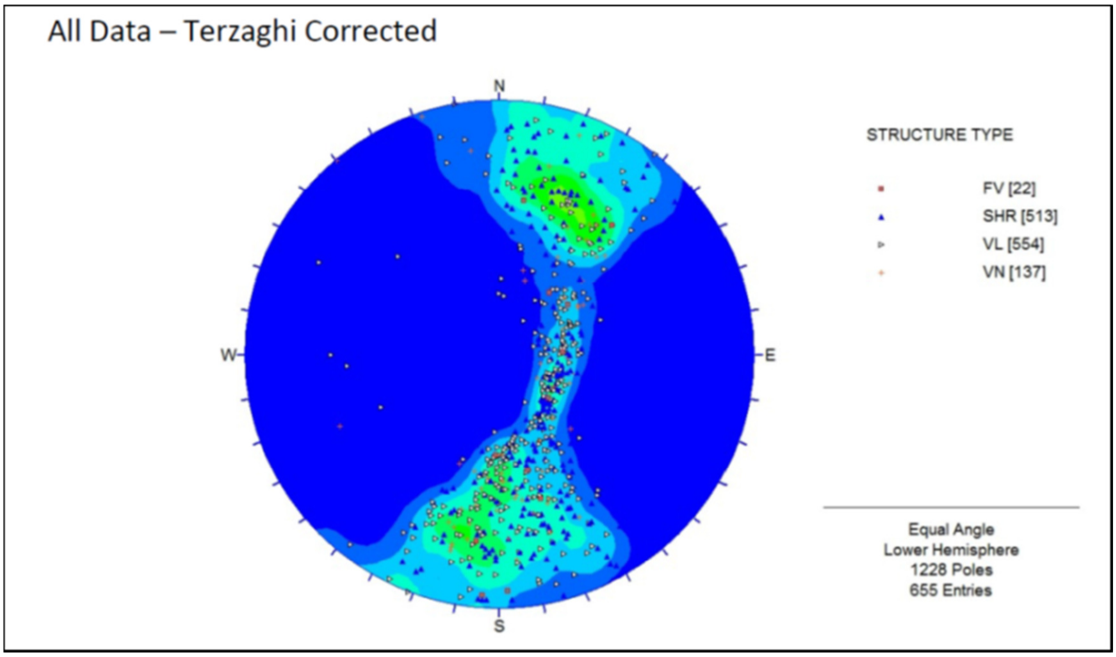


Figure 7.3-1 Equal angle lower hemisphere projection of all structural orientation data.

GCE were also asked to comment upon whether the supplied data can provide and information on rock strength and excavatability. GCE used RQD data and rock strength from the test work of Ir. Alwin Darmawan where by and RQD <50 and rock strength of less than 3 MPa equated to easy ripping (Table 7.3-3).

Mancala notes the considerable variability in the depth from surface for easy ripping, and the small data set involved make drawing conclusion from such results tenuous. However, as the cost implication of ripping vs. drill and blast is potentially significant, Mancala has used the mean of the “depth from surface” after removing to highest and lowest values. This gives an average depth form surface of 35m where by the rock will be easily ripped by a D8 sized tracked dozer. This attribute has been incorporated into the block model for cost and physical reporting.

Hole ID	Downhole depth (m) where strength is up to R3 and RQD<50%	Depth from surface (m) (provided by Mancala)	Percentage of total hole length
DDH1022	18.2	15.9	17%
DDH1023	37	35.8	39%
DDH1025	58.9	53	62%
DDH1026	71.5	69	100%
DDH1036	83	71.6	67%
DDH1037	54	49.3	63%
DDH1042	33.7	30	35%
DDH1124	34.9	29	30%
DDH1131	15.3	12	13%
DDH1132	39.3	29	30%
DDH1138	54.3	47	100%
DDH1141	120	105	100%
DDH1142	25.7	22	21%
DDH1143	39.3	36	77%
DDH1144	27	21	24%

Table 7.3-3 Rock strength and RQD of the structural data set.

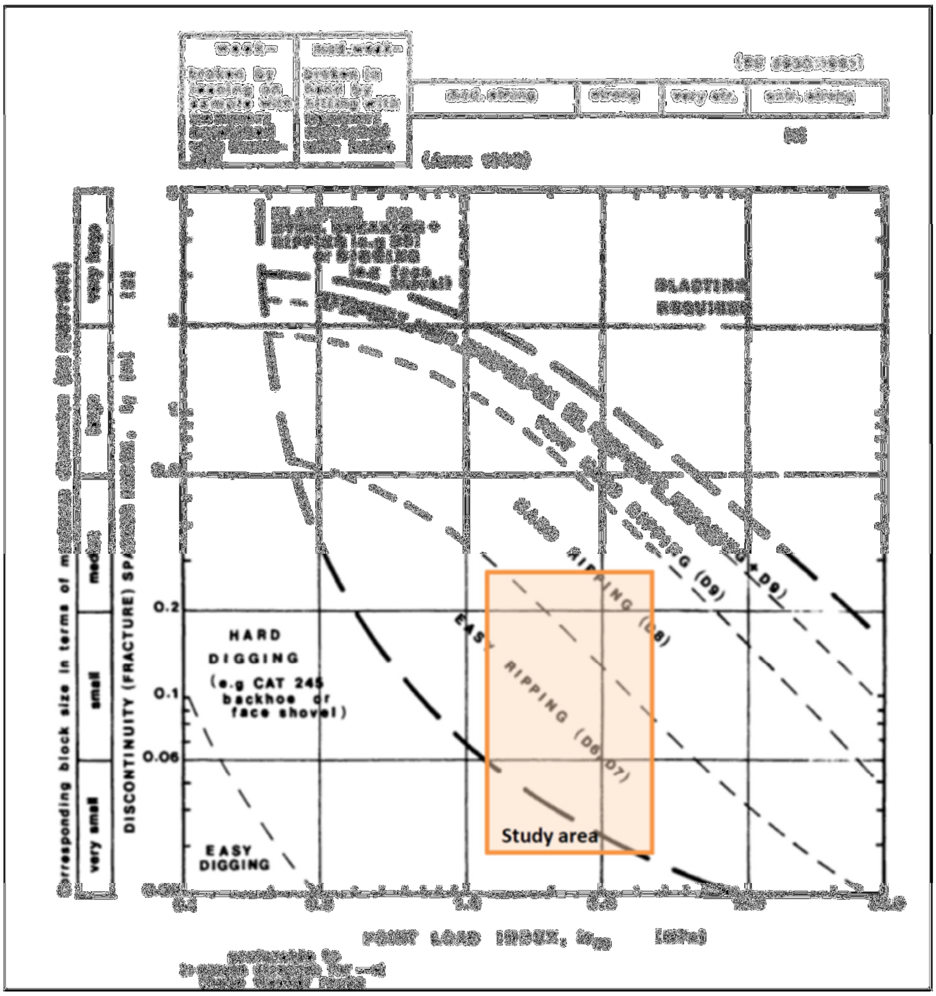


Figure 7.3-2 Excavatability plot using rock strength and fracture spacing

8 Open Cut Geological Data

Geological block model and resource modelling was completed by ASI and SRK 2014. The models supplied to Mancala contain mineralised blocks of variable sizes as listed below:

BM_CIBATU_OK_20140104.csv
BM_CIKADU_OK_20140104.csv
BM_PSM_EXT_OK_20140104.csv
BM_PSM_WN_OK_20140104.csv
BM_PSM_WS_OK_20140104.csv
BM_SEKOLAH_N_OK_20140104.csv
BM_SEKOLAH_S_OK_20140104.csv

These files were combined into a one regularised 5 x 5 x 5 block model with ore allocated as a proportion of the block. The new model was validated against the original model to ensure the accuracy of contained mineralisation was maintained. The variation was less than 0.5%.

8.1 Block Model

The block model created has the attributes shown in Table 8.1-1.

Item	unit	comment
EAST	m	block centroid easting
X Block size	M	5
NORTH	m	block centroid northing
Y Block Size	M	5
ELEV	m	block centroid RL
Z Block size	M	5
Au_ppm	Ppm	Resource gold grade
Au_dil	Ppm	Diluted reserve gold grade
Gold	g	Grams of gold in the block
res_tonnes	NA	Tonnes of mineralised resource in the block
p_ore	NA	% of the block that is mineralised resource
P_dil	%	% of dilution
P_ore_dil_rec	%	% of the block that is resource ore (includes dilution and recovery
P_topo	%	% of block below topography surface
Sched_code	NA	Material type as scheduled
SG	t/m3	Specific gravity
Whittle_density	t/m3	Assignment of tonnes for Whittle
Whittle_code	%	Whittle rock type code
Zone	NA	Zone (character code)

Table 8.1-1 Block model attributes.

8.2 Free Dig Definition

The geotechnical review by GCE determined that mining was unlikely to require blasting from the surface to 35 meters depth. All blocks to a depth of 35m from surface were assigned as free dig.

9 Pit Optimisation

Pit optimisation was carried out on the regularised resource model incorporating mining and processing parameters. The optimisation included all mineral resource categories including the Inferred Resources. Mancala notes that the 2004 and 2012 versions of the JORC code do not allow Reserve reporting from Inferred Resources. Any reporting incorporating Inferred Resources herein are termed Mining Inventory or Production Profile or Production Schedule.

9.1 Pit Optimisation Parameters

The pit optimisation parameters were based on benchmarked costs from recent projects in Indonesia with the capital cost estimates provided by PT WWI.

The pit optimisation parameters are presented in Table 9.1-1.

Parameter	Value	Comment
Wall Slope	40 Degrees	Based on recent slop angle for similar Indonesian oxide project
Ore Mining Cost	$4.00 per tonne	Benchmark cost from recent feasibility studies
Waste	$4.00 per tonne	Benchmark cost from recent feasibility studies
Mining cost adjustment factor	$0	0 for this initial scope to be modified to allow for haulage distance and mining depth in subsequent studies
Mine dilution	10%	Experience based estimate
Mine recovery	90%	Experience based estimate
Processing Cost	$44 per tonne ($21 process and $23 G&A)	SRK Report
Processing Recovery	90%	SRK Report
Sell/Realisation costs	$42 per oz	Indonesian royalty change of 3.5%
Metal Prices	$1200.00 per oz	Conservative estimate due to declining prices. $1551 3 year average (2011-2013) $1412 1 year average (2013)
Capital	$90 Million initial zero sustaining capital	Estimate from SRK Report
Discount Rate	10%	
Limits	500,000 tpa processing 5,000,000 tpa mining	SRK Report Estimate of stripping ratio

Table 9.1-1 Pit optimisation parameters.

After the optimisation had been completed, it was identified that the estimated dilution should probably have been higher and the mining recovery also higher. The selling cost should also have been higher to include refining and security of doré bars. Given the conservative gold price used ($1,200/oz) this is not considered material at this level of study.

9.2 Optimisation Results

The first pass Whittle optimisation shows that the Ciemas deposits can be mined economically by open cut methods. The following graphs show the result of the optimisation.

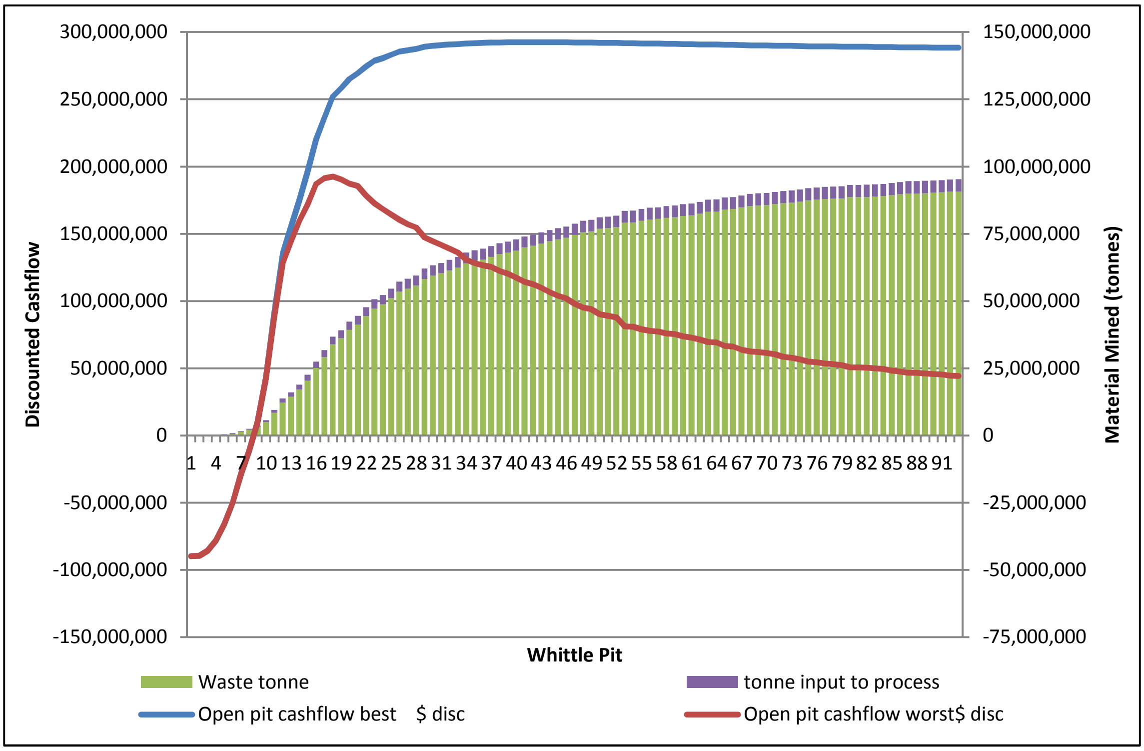


Figure 9.2-1 Optimisation shell results.

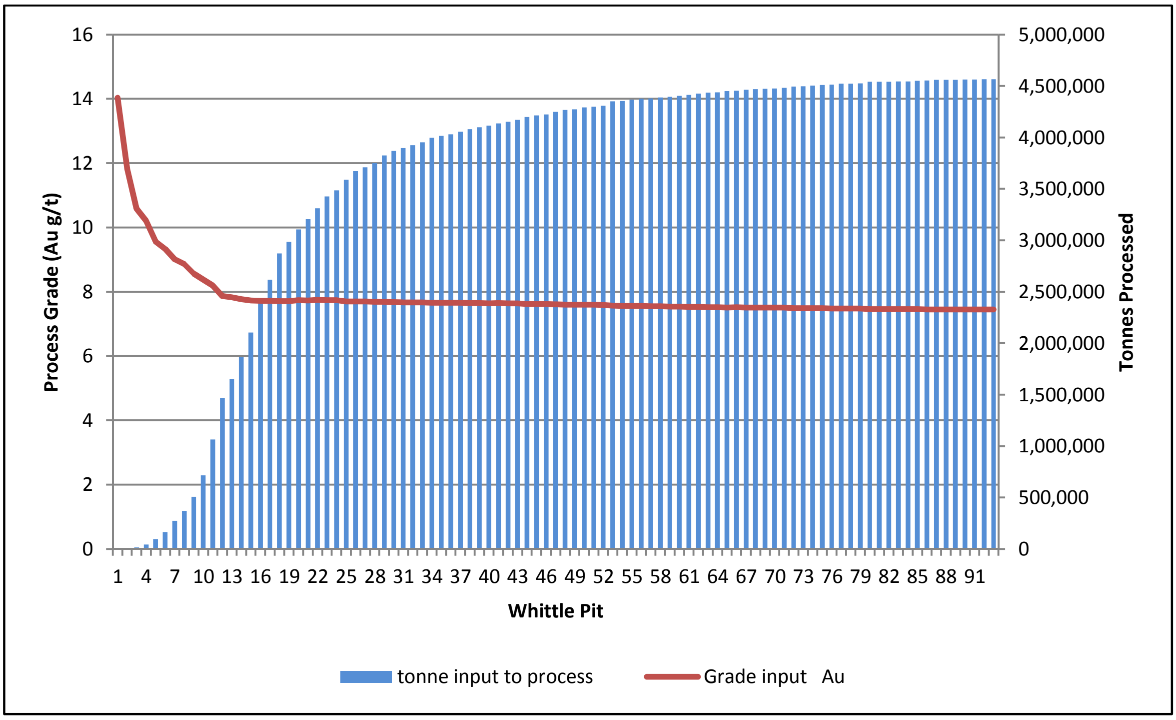


Figure 9.2-2 Optimisation shell tonnes input to process.

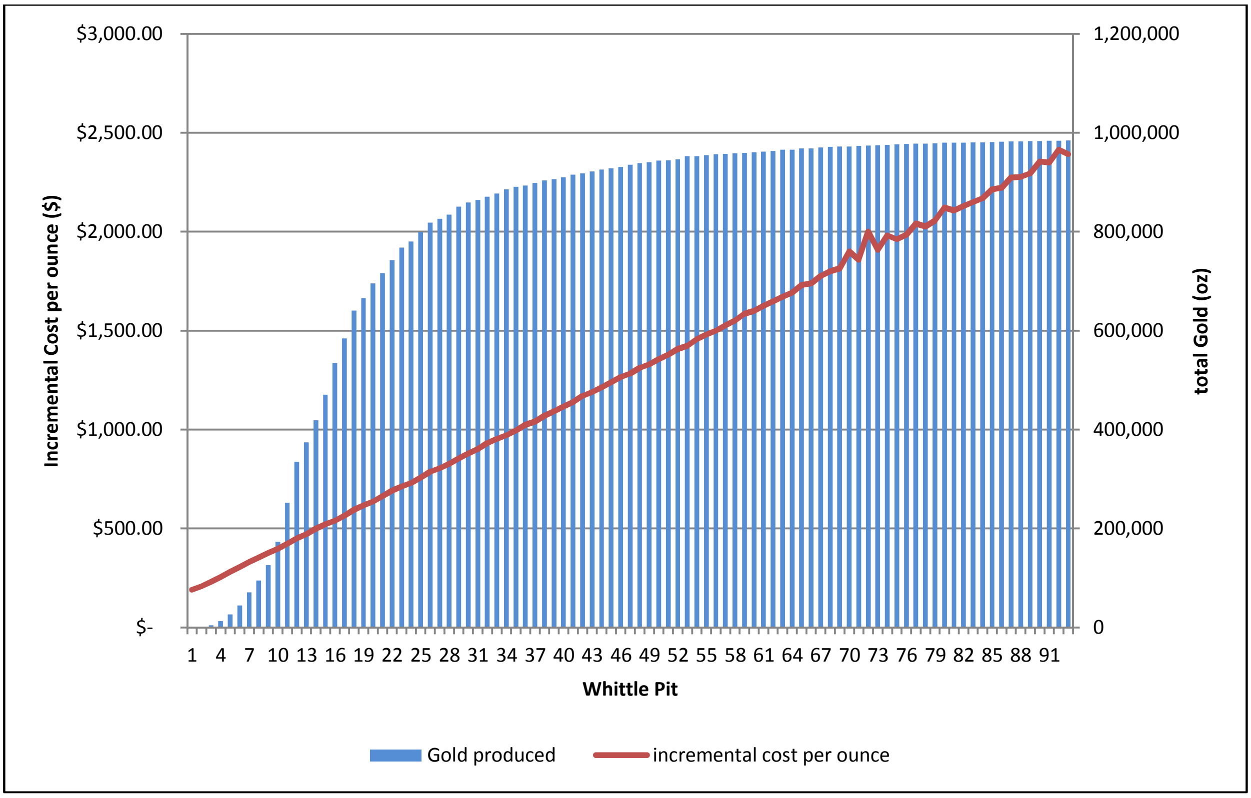


Figure 9.2-3 Gold produced for increasing optimisation shells and the cost per ounce.

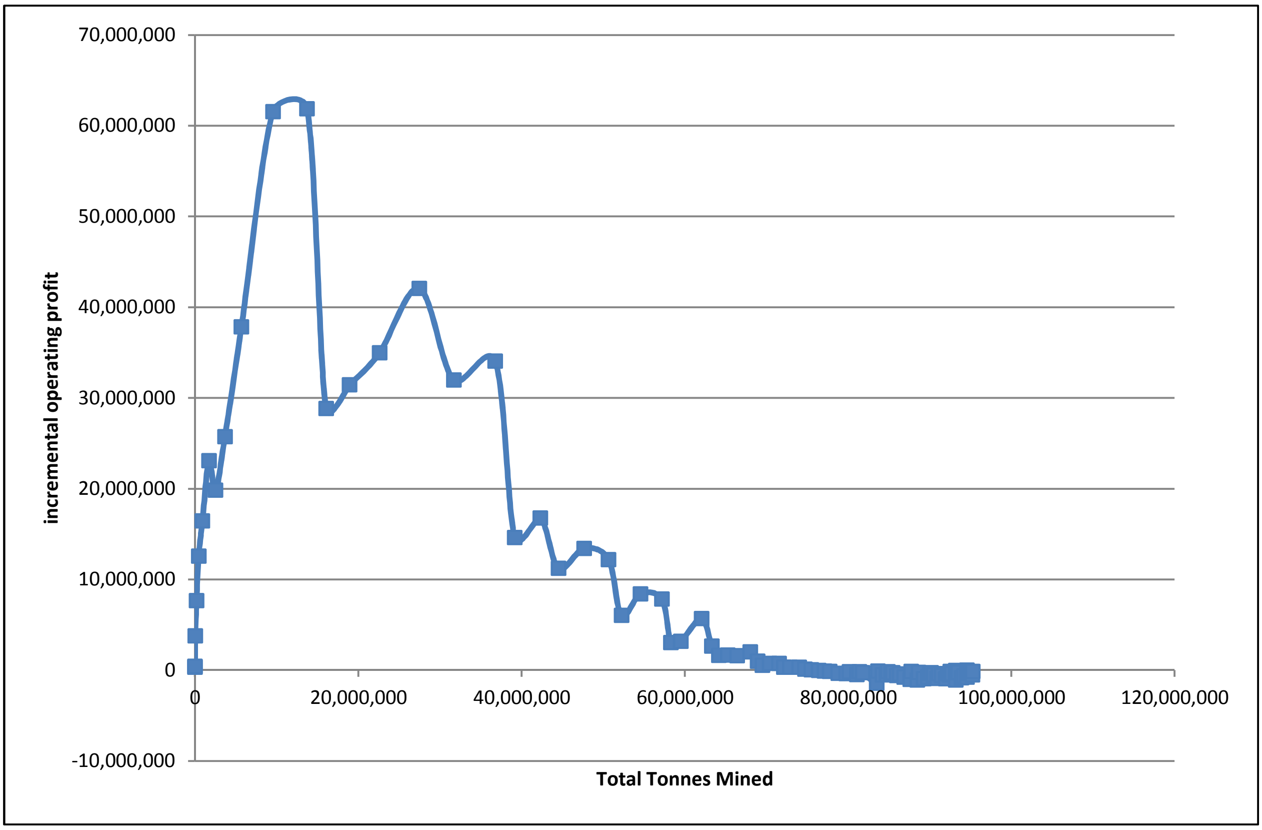


Figure 9.2-4 Incremental profit for increasing tonnes mined.

Based on the large decline in marginal profit from mining 36 million tonnes to mining 39 million tonnes (Optimisation Shell 18 to Optimisation Shell 19 shown in Figure 9.2-4) and the location of Pit Shell 18 near the crest of the NPV curve (Figure 9.2-1), it is considered that Whittle pit 18 represents the best pit shell option from which to design detailed open cuts. Pit 18 has the following table of physical attributes.

Waste Tonnes	Ore Tonnes	Au (g/t)	Gold produced (oz)	Mine life at 500ktpa
33,900,000	2,900,000	7.7	641,000	6 Years

Table 9.2-1 Optimisation Shell 18 contained material.

Mining Cost	Processing Cost	Selling Cost	Revenue	Operating Profit
-147,100,000	-120,700,000	-27,000,000	768,800,000	474,100,000

Note: Operating profit excludes initial capital, rounding will cause summation errors

Table 9.2-2 Optimisation shell 18 financials.

While subject to more detailed design, scheduling and financial modelling if the estimated $90 million capital and a 10% discount rate is applied, an open cut mining option is likely to produce an NPV of $190 - $220 million.

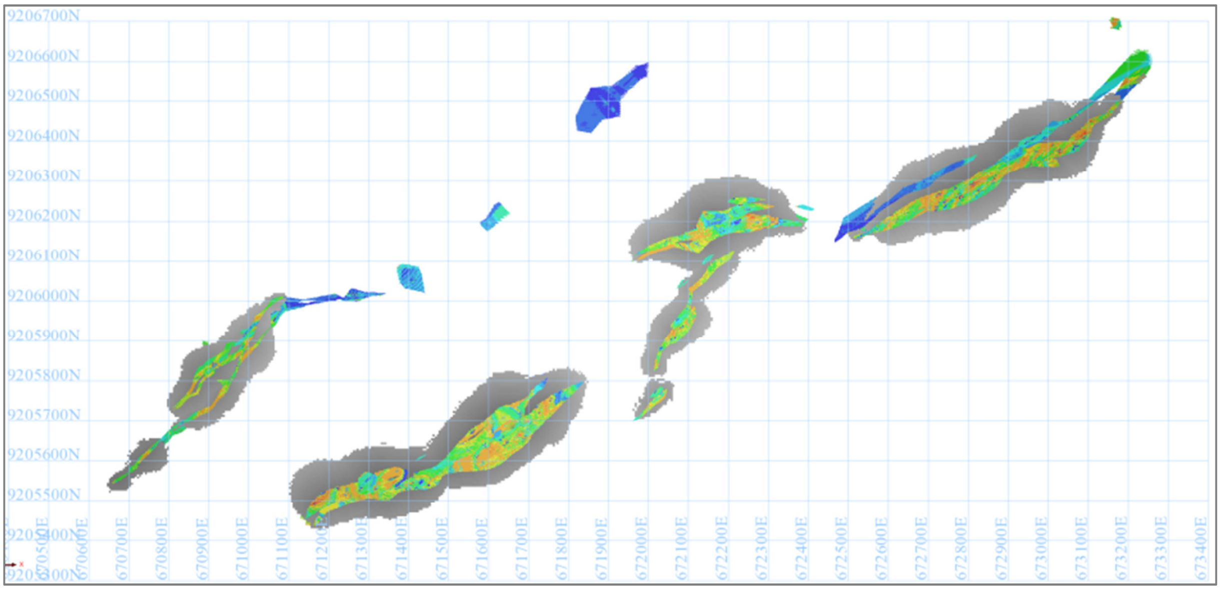


Figure 9.2-5 Pit 18 and ore lodes. Grid spacing 100m.

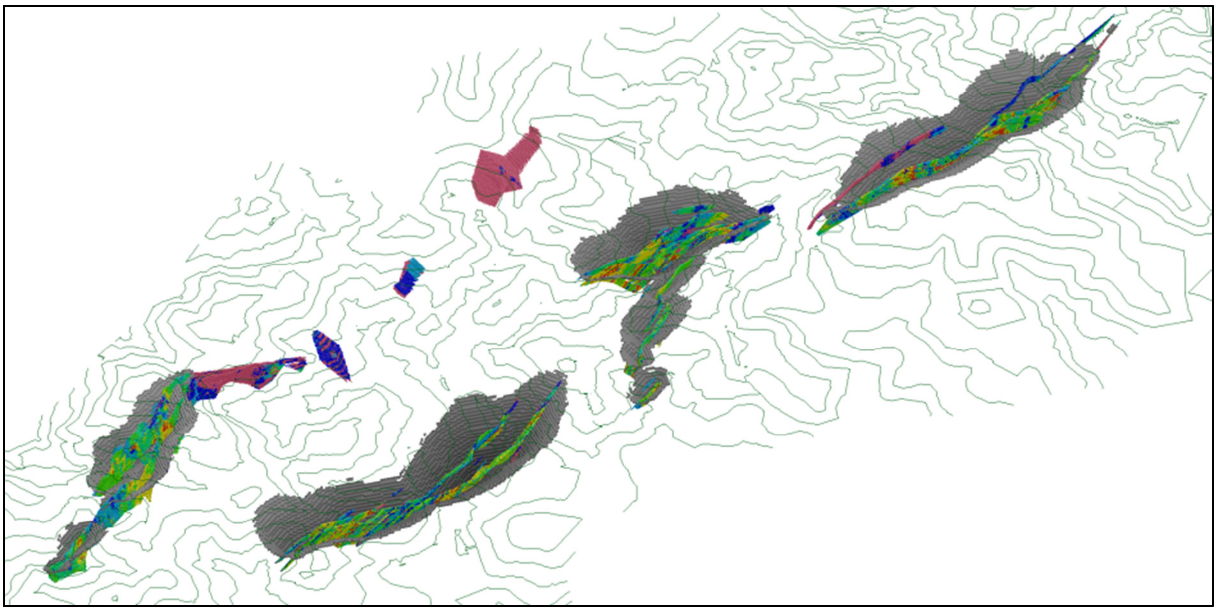


Figure 9.2-6 Pit 18 and lodes isometric view to the north.

9.3 RECOMMENDATION

As it is likely an open cut mining method will be profitable, further optimisation studies incorporating revised input parameters should be completed followed by detailed design work. Based on the initial optimisation results herein, mining of approximately 6 million tonnes per annum (ore and waste) is likely to be required.

10 Mining Dilution and Recovery

Dilution has been estimated based on the proportion of waste in each block of the block model. Within the regularised 5 x 5 x 5 mining block model, each block contains an attribute specifying the percentage of ore that the block contains. This attribute was used to estimate the dilution. It has been assumed that the less ore a block contained the greater the dilution would be when it was mined. Based on operational mining experience and they type of geology the following has been estimated:

- All blocks would have at least 5% dilution even if they were all ore;
- Additional dilution would be equal to half the percentage of waste in a block. Thus if a block contained 80% waste then additional dilution would be 40%; and
- Due to the nature of the geology it has been assumed that there is a hard contact between ore and waste and that the waste material contained no gold. Thus dilution was added at zero grade.

Recovery was estimated to be a consistent 95% for all diluted ore.

10.1 Dilution and Recovery Impact

Table 10.1-1 depicts the average results of the dilution and recovery for the deposit.

Original Resources	Volume	Tonnes	Au Ppm	Gold (oz)
Pasir Manggu	70,000	200,000	7.8	49,000
Cikadu	410,000	1,110,000	8.7	311,000
Sekolah	240,000	650,000	8.0	167,000
Cibatu	250,000	670,000	8.6	185,000
Grand Total	**970,000**	**2,620,000**	**8.5**	**712,000**
Mining diluted and recovered	**Volume dil rec**	**Tonnes dil rec**	**Au dil**	
Pasir Manggu	93,000	250,00	5.8	47,000
Cikadu	500,000	1,350,000	6.8	295,000
Sekolah	300,000	810,000	6.1	159,000
Cibatu	296,000	800,000	6.9	175,000
Grand Total	**1,190,000**	**3,210,000**	**6.6**	**677,000**
% Differences				
Pasir Manggu	127%	127%	75%	96%
Cikadu	122%	122%	78%	95%
Sekolah	126%	126%	76%	96%
Cibatu	119%	119%	80%	95%
Grand Total	**122%**	**122%**	**78%**	**95%**

Table 10.1-1 Dilution and recovery of the Mineral Resources. Rounding will cause summation errors.

The Mining Inventory to be exploited by open cut methods is estimated as being:

3,210,000 tonnes at 6.6g/t Au for 677,000 oz gold

The Mining Inventory does not constitute an Ore Reserve. The Mining Inventory has been compiled based on low level technical and economic assessments, and is insufficient to support estimation of Ore Reserves or to provide assurance of an economic development case at this stage, or to provide certainty that the conclusions of the Scoping Study will be realised.

11 CUT OFF CALCULATIONS

11.1 CUT OF GRADES

The Cut Off Grade (COG) calculations are shown in Table 11.1-1. Dilution and recovery is 0% and 100% as it has been incorporated in the block model. Based on the cut-off grade calculations it has been elected to treat 3.0 g/t and above as direct feed ore and 1.0 – 3.0 g/t as low grade ore. This ore is stockpiled for processing at the end of mine life.

Mining Cut Off			Processing Cut Off			Low Grade Stockpile Cut Off		
Tonnes		1.0	Tonnes		1.0	Tonnes		1.0
Block Au Grade	g/t	2.81	Block Au Grade	g/t	1.25	Block Au Grade	g/t	1.01
Mining			Mining			Mining		
Waste	$/tonne	4.00	Waste	$/tonne		Waste	$/tonne	4.00
Waste tonne		12.0	Waste tonne			Waste tonne		12.0
Mining Cost	$/tonne	4.00	Mining Cost	$/tonne		Reclaim Cost	$/tonne	0.5
MCAF	$/tonne	0.00	MCAF	$/tonne		MCAF	$/tonne	0.00
Density	t/BCM	2.7	Density	t/BCM	2.7	Density	t/BCM	2.7
Dilution		0%	Dilution		0%	Dilution		0%
Recovery		100%	Recovery		100%	Recovery		98%
Mined Tonnes		1.0	Mined Tonnes		1.0	Mined Tonnes		1.0
Actual cost	$US	52.0	Actual cost	$US	0	Actual cost	$US	0.5
Mined Au Grade	g/t	2.81	Mined Au Grade	g/t	1.25	Mined Au Grade	g/t	1.03
Mined Gold	grams	2.81	Mined Gold	grams	1.25	Mined Gold	grams	0.99
Processing			Processing			Processing		
Processed Tonnes	t	1.00	Processed Tonnes	t	1.00	Processed Tonnes	t	0.98
Processed grade	g/t	2.81	Processed grade	g/t	1.25	Processed grade	g/t	0.99
Operating cost	$US	42	Operating cost	$US	42	Operating cost	$US	42
Capital Cost	$US		Capital Cost	$US		Capital Cost	$US	
Process recovery Au	%	90	Process recovery Au	%	90	Process recovery Au	%	90
Actual Cost	$US	42	Actual Cost	SUS	42	Actual Cost	SUS	41.16
Recovered Gold	grams	2.52	Recovered Gold	grams	1.13			
Selling			Selling					
Realisation Cost au	$/g	1.35	Realisation Cost au	$/g	1.35			
Actual selling cost au	$US	3.41	Actual selling cost au	$US	1.52			
Total Cost	$	97.41	Total Cost	$	43.52			
Revenue			Revenue					
Gold Price	$/oz	1,200	Gold Price	$/oz	1,200			
Revenue	$	97.41	Revenue	$	43.52			
Profit	$	0.00	**Profit**	$	0.00			

Table 11.1-1 Cut Off Grade (COG) calculations.

12 Open Cut Design

Based on the pit optimisation exercise, Shell 18 was selected as a basis for the open cut mine design.

12.1 Design Criteria

The following mine design criteria was used.

12.1.1 Mining Method

The mining method will be open cut using excavators and articulated dump trucks.

Excavation equipment is assumed to be:

- 90 tonne excavators mining waste only, loading a fleet of 40 tonne articulated dump trucks; and
- 45 tonne excavators loading ore and waste into 40 tonne articulated trucks.

12.1.2 Open Cut Wall Angles

The proposed bench and batter design is shown in Table 12.1-1. The parameters were based on experience from other similar open cut projects and advice from Ground Control Engineering.

Parameter	Value/Units
Free Dig Bench Height	10 meters
Free Dig Berm Width	5 Meters
Free Dig Face Angle	45 degrees
Drill and Blast Bench Height	15 meters
Drill and Blast Berm Width	8 Meters
Drill and Blast Face Angle	65 degrees
Overall Maximum Slope	45 degrees
Duel Road Width	16 meters
Single Road Width	10 meters
Road Slope	1:10 (vertical: horizontal)
Minimum Working Area	1,500 m2

Table 12.1-1 Open cut design parameters.

12.1.3 Bench Configuration

The configuration of 5m interim benches with a final bench height of 15m was selected as it provides the best combination to meet geotechnical and/or operational requirements and is the maximum bench height permitted under Indonesian regulations.

The flitch height of 5m was based on the following assessment:

- Drill and blast at 5m will provide good grade control and ability to minimize dilution during blasting; and
- 5m bench will swell to approximately 6m with blasting allowing optimum mining depth

of two by 3 meter passes with a Cat 90 tonne (or similar) excavator with a 3m mass excavation stick.

12.2 Final Open Cut Designs

The final open cut design is shown in plan in Figure 12.2-1 and isometric in Figure 12.2-2. The images show the main open cut areas.

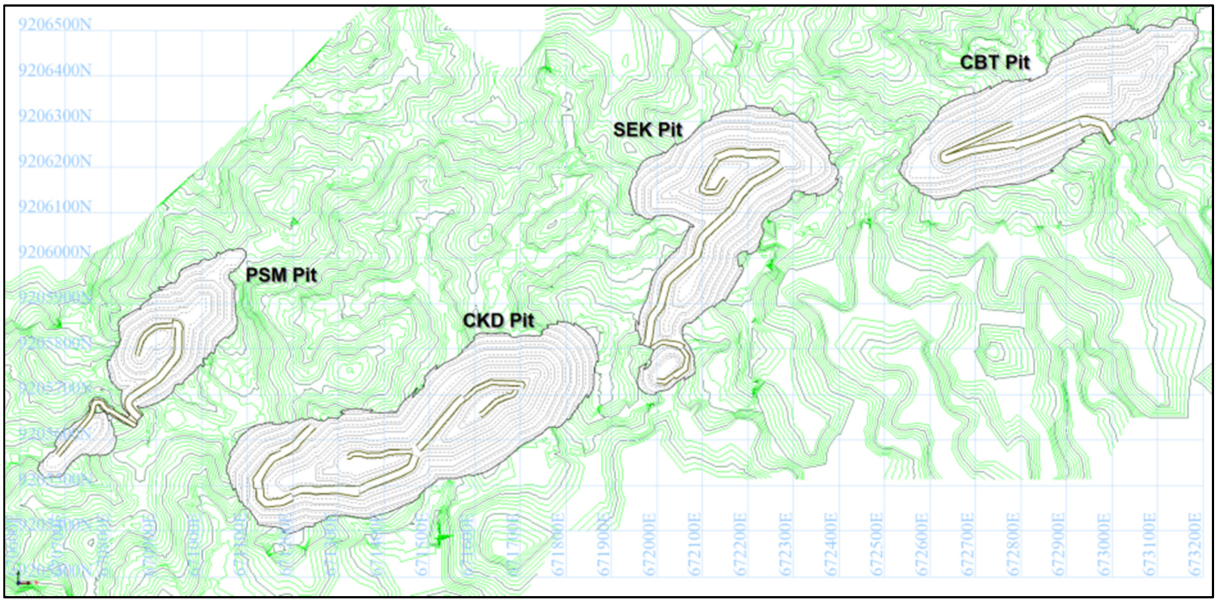


Figure 12.2-1 36Mtpa open cut design plan view.

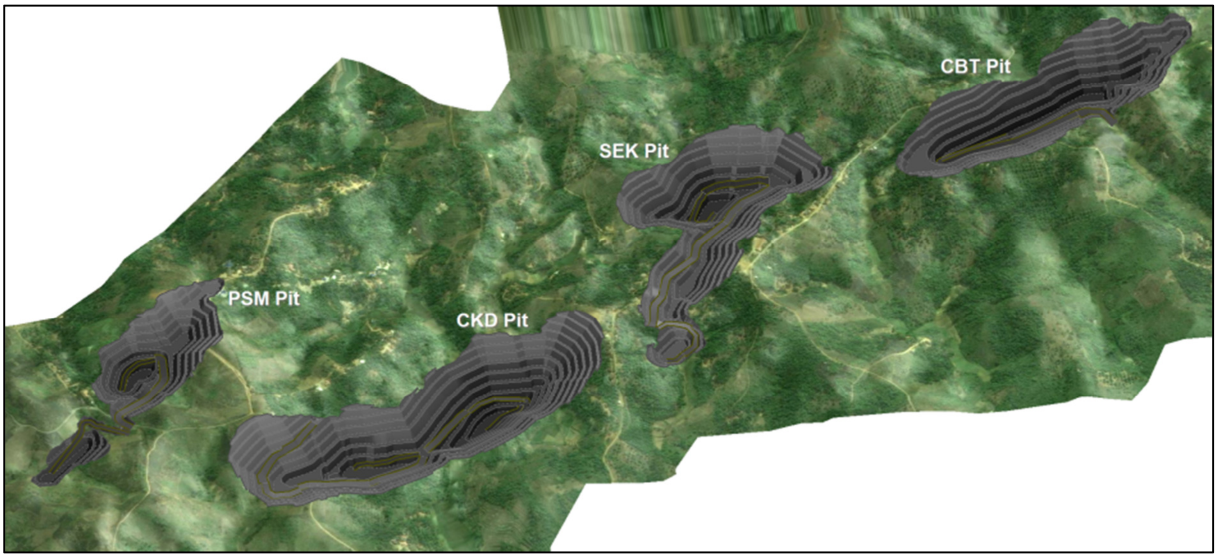


Figure 12.2-2 36Mtpa open cut design isometric view to north (without waste dumps).

The open cut design contains the physicals shown in Table 12.2-1 and the grade tonnage curve for the design is shown in Figure 12.2-3.

	Ore (t)	Average Au (g/t)	Au (oz)	Waste (t)	Strip Ratio (t:t)	Depth (RL)	Footprint (Ha)
PSM	250,500	5.8	47,000	3,110,000	12.4	460	7.4
CKD	1,350,000	6.8	295,000	17,010,000	12.6	385	18.4
SEK	810,000	6.1	159,00	8,360,000	10.2	420	13.3
CBT	800,000	6.9	175,00	8,630,000	10.8	450	12.4
Total	**3,210,000**	**6.6**	**677,000**	**37,110,000**	**11.6**	**NA**	**51.5**

Table 12.2-1 Open cut physicals. Rounding will cause summation errors.

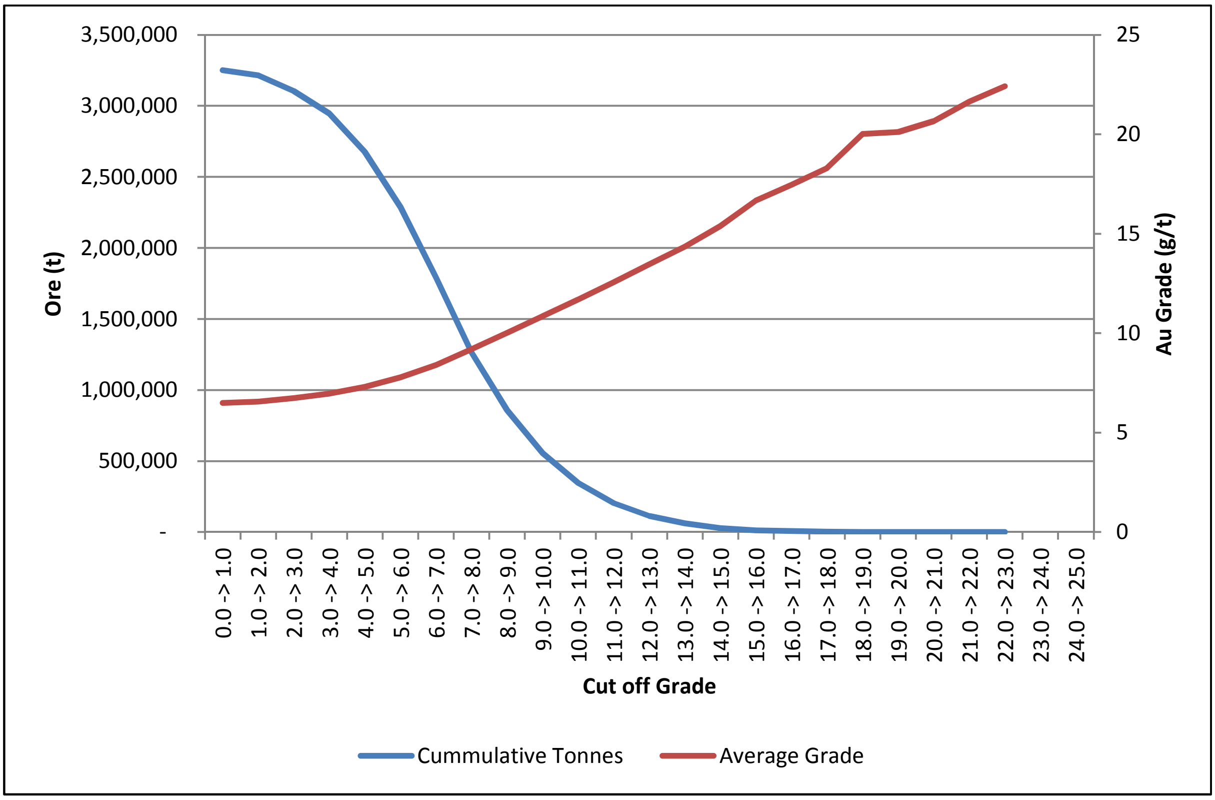


Figure 12.2-3 Grade tonnage curve for material contained within the designed open cuts.

The Mineral Resources contained in the open cut designs are shown in Table 12.2-2. The open cut mine designs incorporate 57% of the defined Mineral Resources (including Inferred Resources). The remaining Mineral Resources could be recovered by underground mining methods as described in Section 19.

Resource Class and Open Cut	Tonnes	Au (g/t)	Au (oz)	% tonnes of total Resource
Total Measured Resources	**100,000**	**8.1**	**23,000**	**2%**
PSM	100,000	8.1	23,000	2%
Total Indicated Resources	**1,850,000**	**8.5**	**504,000**	**40%**
PSM	90,000	8.4	23,000	2%
CKD	890,000	8.8	252,000	19%
SEK	460,000	7.9	118,000	10%
CBT	420,000	8.9	119,000	9%
Total Inferred Resources	**670,000**	**8.3**	**178,000**	**14%**
PSM	10,000	7.7	3,000	0%
CKD	220,000	8.3	59,000	5%
SEK	180,000	8.4	49,000	4%
CBT	250,000	8.2	66,000	5%
Total Resources	**2,620,000**	**8.5**	**712,000**	**57%**

Table 12.2-2 Mineral Resources contained within the open cut designs. Rounding will cause summation errors.

The open cut mined ore is a combination of oxide ore and primary ore. The percentage slipt between oxide and primary ore is shown in Figure 12.2-4.

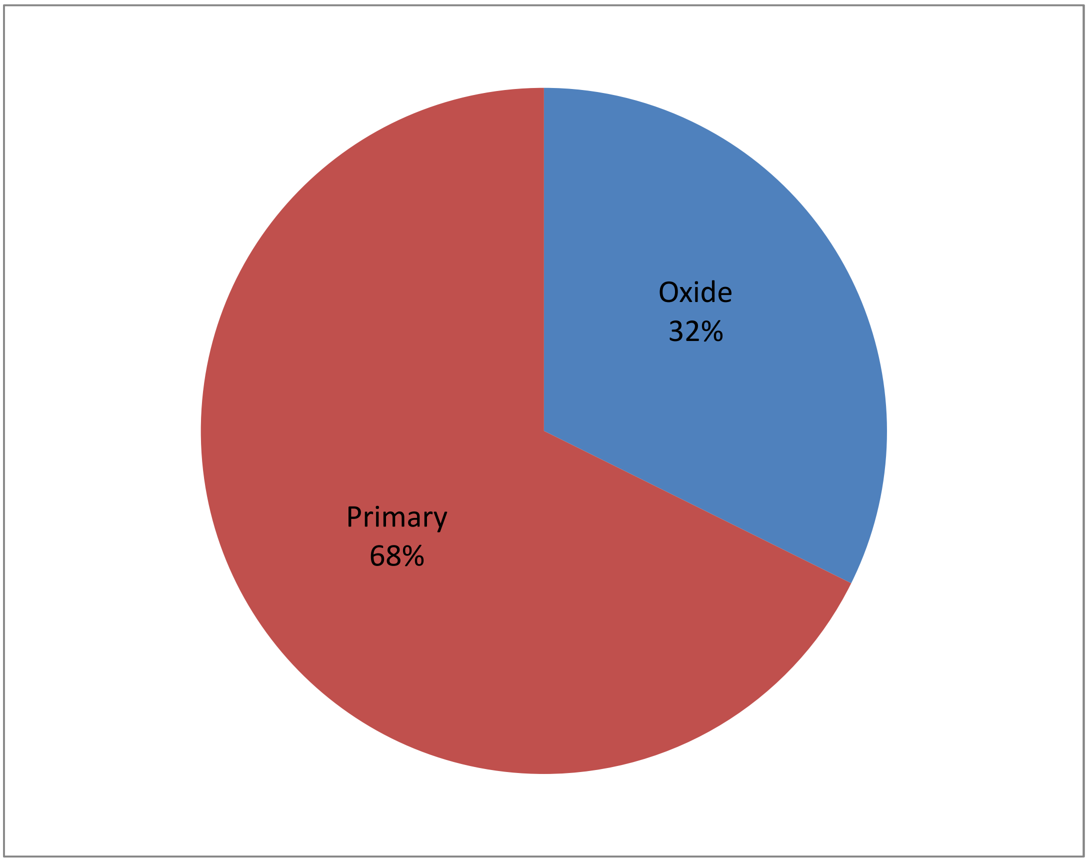


Figure 12.2-4 – Mix of oxide and primary ore in the open cut designs

12.2.1 Pasir Manggu (PSM) Open Cut

PSM is the western most open cut and the most comprehensively defined by drilling. The open cut is the smallest and the lowest grade of all the designs, but due to having the highest resources confidence level it is scheduled to be the first open cut mined. The physicals for the open cut are shown in Table 12.2-3 and images with ore coloured by grade in Figure 12.2-5.

Item	Units	Value
Total Movement	BCM	1,270,000
Waste	BCM	1,180,000
Ore	BCM	90,000
Waste Free Dig	BCM	1,130,000
Waste Free Dig	tonnes	2,980,000
Waste Drill Blast	BCM	50,000
Waste Drill Blast	tonnes	130,000
Ore Free Dig	BCM	80,000
Ore Free Dig	tonnes	210,000
Ore Drill Blast	BCM	10,000
Ore Drill Blast	tonnes	20,000
Ore Au	grams	1,430,000
Ore Au (g/t)	grams/tonne	6.1
Sub-Grade Free Dig	BCM	10,000
Sub-Grade Free Dig	tonnes	20,000
Sub-Grade Drill Blast	BCM	0
Sub-Grade Drill Blast	tonnes	0
Sub-Grade Au	grams	30,000
Sub-Grade Au	grams/tonne	2.3

Table 12.2-3 PSM open cut physicals. Rounding will cause summation errors.

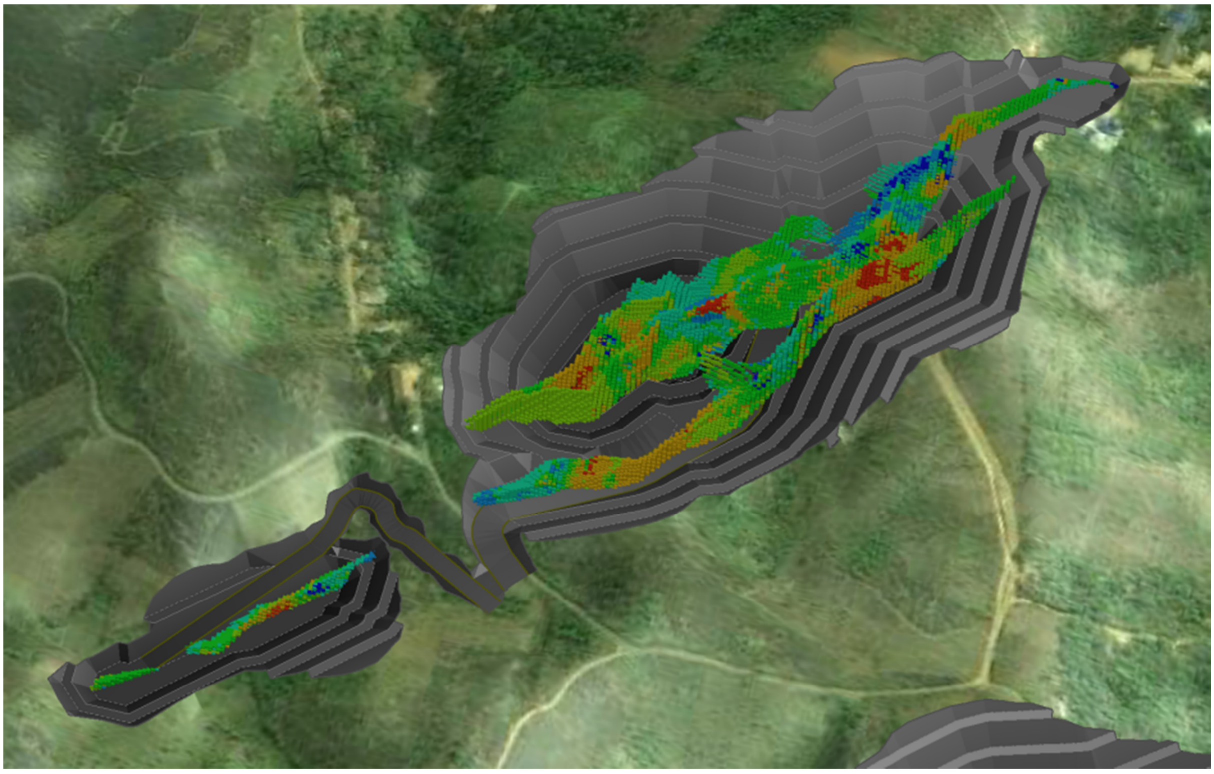


Figure 12.2-5 PSM open cut design and resource model.

12.2.2 Cikadu (CKD) Open Cut

CKD is the largest of the four open cuts. It commences in year two of the schedule and continues being mined until month five of year seven. As CKD open cut is mined continuously from year two onwards, there will always be two open cuts in operation. Due to the open cut's size and the high stripping ratio of the upper benches, staging and a cut back should be considered in future work. The ramp exit from could also be moved to the south wall in the next design phase to reduce waste haulage distances.

Item	Units	Value
Total Movement	BCM	6,890,000
Waste	BCM	6,390,000
Ore	BCM	500,000
Waste Free Dig	BCM	4,400,000
Waste Free Dig	tonnes	11,620,000
Waste Drill Blast	BCM	1,990,000
Waste Drill Blast	tonnes	5,380,000
Ore Free Dig	BCM	190,000
Ore Free Dig	tonnes	510,000
Ore Drill Blast	BCM	280,000
Ore Drill Blast	tonnes	770,000
Ore Au	grams	9,020,000
Ore Au	grams/tonne	7.1
LG Free Dig	BCM	20,000
LG Free Dig	tonnes	40,000
LG Drill Blast	BCM	10,000
LG Drill Blast	tonnes	30,000
LG Au	grams	170,000

Table 12.2-4 CKD open cut physicals. Rounding will cause summation errors.

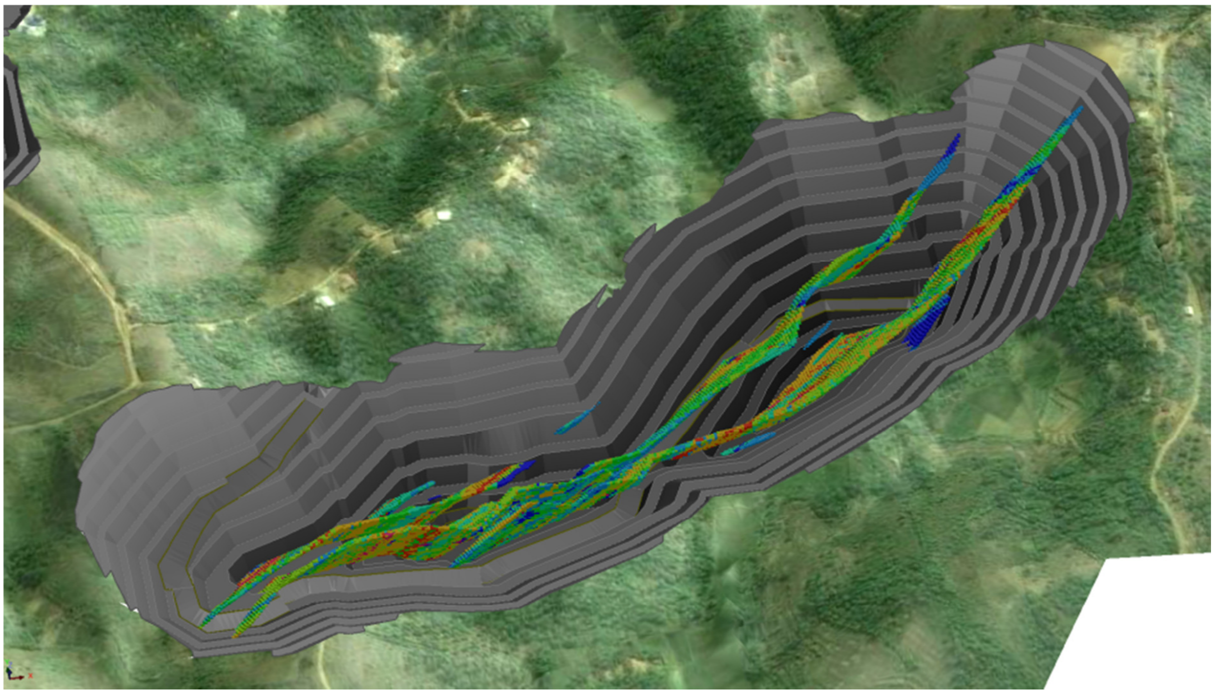


Figure 12.2-6 Isometric view CKD open cut design and resource model.

12.2.3 Sekolah (SEK) Open Cut

The SEK is the second open cut to commence. It commences with waste stripping in month 9 of year 1. The open cut is in the central east of the project area. Access is via a north/south running ramp along strike of the ore lens. Water management strategies will all need to be completed by the commencement of this open cut as a water course is intersected and the haulage route will transect a water course.

Item	Units	Value
Total Movement	BCM	3,430,000
Waste	BCM	3,130,000
Ore	BCM	300,000
Waste Free Dig	BCM	2,580,000
Waste Free Dig	tonnes	6,860,000
Waste Drill Blast	BCM	550,000
Waste Drill Blast	tonnes	1,490,000
Ore Free Dig	BCM	160,000
Ore Free Dig	tonnes	440,000
Ore Drill Blast	BCM	110,000
Ore Drill Blast	tonnes	300,000
Ore Au	grams	4,800,000
Ore Au	grams/tonne	6.5
LG Free Dig	BCM	20,000
LG Free Dig	tonnes	50,000
LG Drill Blast	BCM	10,000
LG Drill Blast	tonnes	20,000
LG Au	grams	160,000
LG Au	grams/tonne	2.3

Table 12.2-5 SEK open cut physicals. Rounding will cause summation errors.

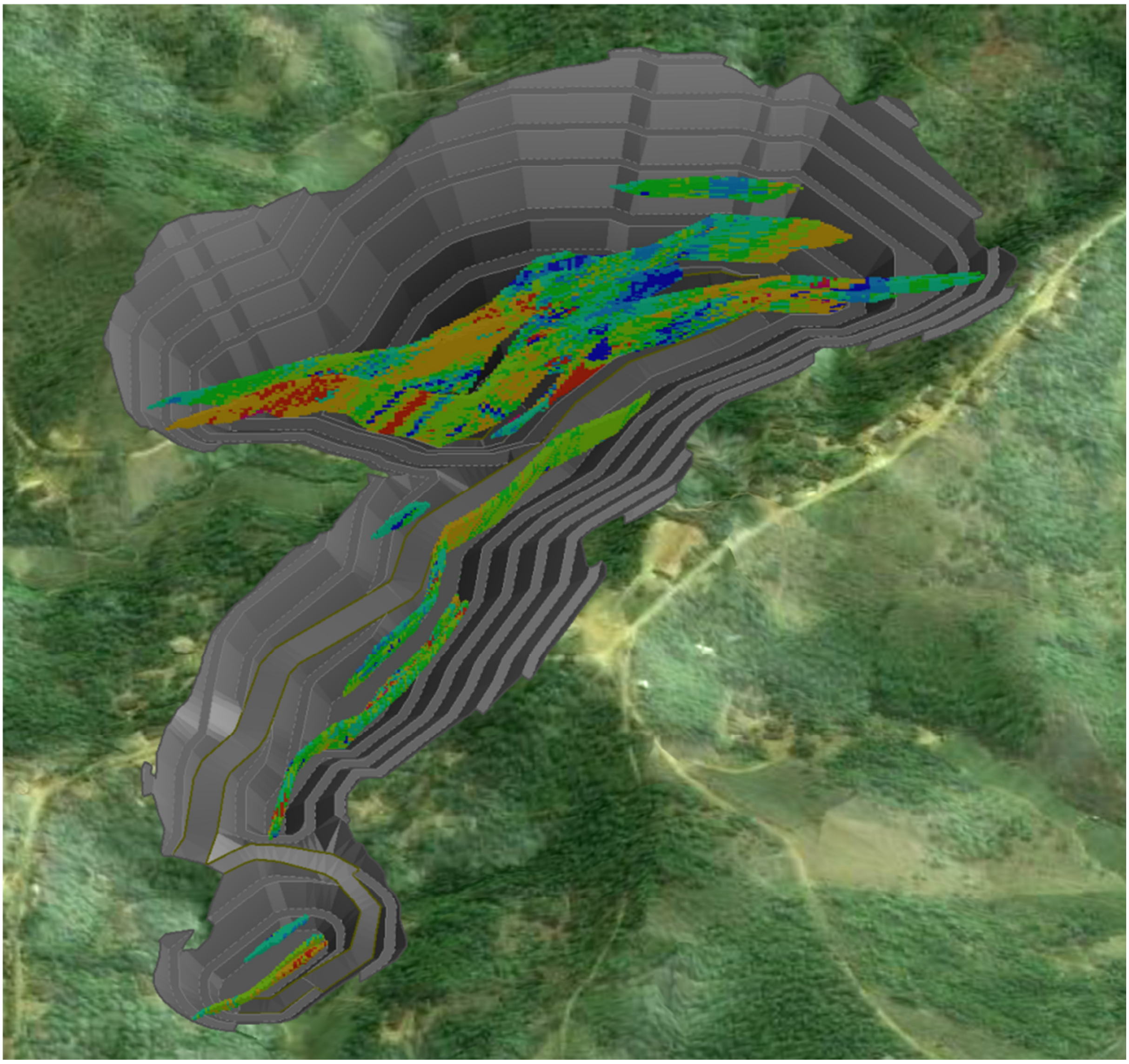


Figure 12.2-7 – Isometric view of SEK open cut design and resource model.

12.2.4 Cibatu (CBT) Open Cut

CBT is the eastern most open cut and the furthest from the proposed plant location. The CBT open cut is mined last in the schedule due to its distance from the proposed plant location and lower drilling density.

Item	Units	Value
Total Movement	BCM	3,540,000
Waste	BCM	3,240,000
Ore	BCM	300,000
Waste Free Dig	BCM	2,650,000
Waste Free Dig	tonnes	7,030,000
Waste Drill Blast	BCM	600,000
Waste Drill Blast	tonnes	1,610,000
Ore Free Dig	BCM	140,000
Ore Free Dig	tonnes	380,000
Ore Drill Blast	BCM	110,000
Ore Drill Blast	tonnes	310,000
Ore Au	grams	5,230,000
Ore Au	grams/tonne	7.6
LG Free Dig	BCM	40,000
LG Free Dig	tonnes	100,000
LG Drill Blast	BCM	0
LG Drill Blast	tonnes	10,000
LG Au	grams	220,000
LG Au	grams/tonne	2.0

Table 12.2-6 CBT open cut physicals. Rounding will cause summation errors.

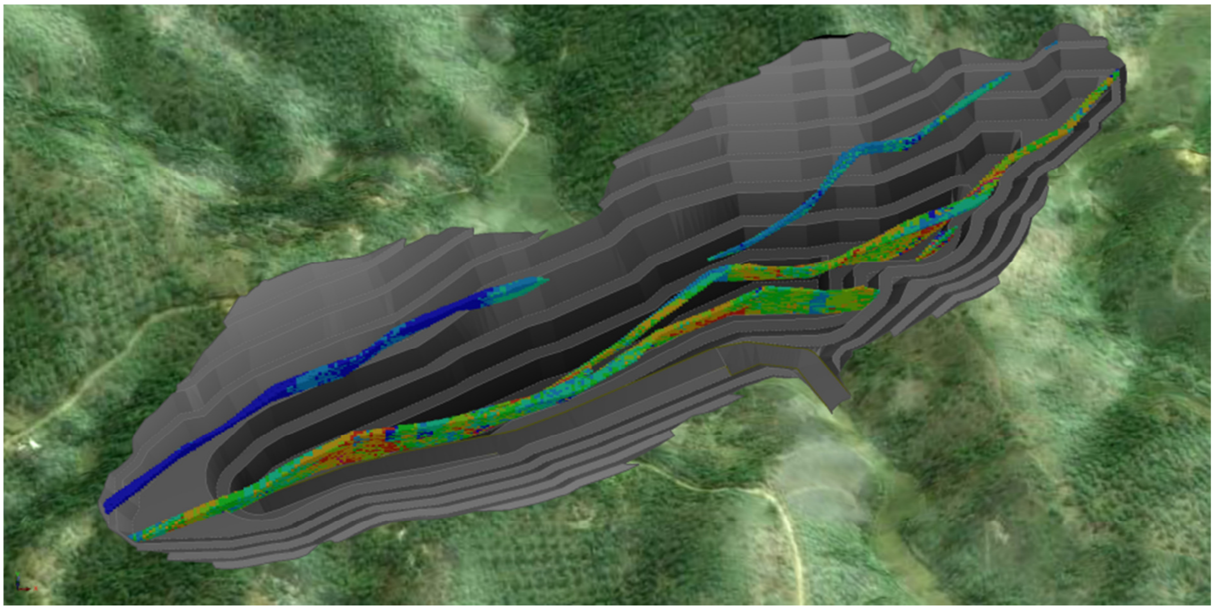


Figure 12.2-8 Isometric view of CBT open cut and resource model.

12.3 Reconciliation to Pit Optimisation

The reconciliation of the designed open cuts against the optimisation shell No. 18 is shown in Table 12.3-1 and Table 12.3-2.

The difference between the two is:

- An increase in the waste movement due to the shallower wall angle in free dig material;
- The need for achieving minimum working room and the ramps in the open cuts;
- An increase in ore tonnes due to increased dilution compared to original optimisation parameters; and
- Decrease in contained gold ounces due to loss of impractical ore in the base of the optimisation pits.

	Ore (Mt)	Average Au (g/t)	Au contained (koz)	Waste (Mt)	Strip Ratio (t:t)
Optimisation Shell	2.90	7.7	719	33.9	11.7
Designed Open Cut	3.21	6.6	677	37.1	11.6

Table 12.3-1 Physicals of the optimised pit shell 18 and the final open cut design.

	Ore (Mt)	Difference Au (koz)	Waste (Mt)
Difference	0.31	-42	3.2
% Difference	10.7%	-5.9%	9.5%

Table 12.3-2 Difference between optimisation pit shell 18 and the final open cut design.

12.4 Remaining Resources

There are resources below and adjacent to the designed open cuts which are not extracted. These resources could be mined by underground methods as conceptually presented in Section 19 of this report. The remaining Mineral Resources outside the open cuts are presented in Table 12.4-1 and as a grade tonnage analysis in Table 12.4-2 and Figure 12.4-1.

Resource Class and Lode	Tonnes	Au (g/t)	Au (oz)
Total Measured Resources	**20,000**	**7.7**	**10,000**
PSM Lodes	20,000	7.7	10,000
Total Indicated Resources	**1,070,000**	**9.3**	**320,000**
PSM Lodes	360,000	7.3	86,000
CKD Lodes	210,000	10.0	69,000
SEK Lodes	250,000	11.7	92,000
CBT Lodes	240,000	9.5	73,000
Total Inferred Resources	**930,000**	**7.2**	**220,000**
PSM Lodes	260,000	3.6	30,000
CKD Lodes	140,000	8.7	38,000
SEK Lodes	120,000	9.0	34,000
CBT Lodes	420,000	8.4	113,000
Total Resources	**2,020,000**	**8.3**	**540,000**

Table 12.4-1 Remaining unmined (via. open cut) resources at Au 1g/t cut off. Rounding will cause summation errors.

Cut Off in Au g/t	Cumulative Tonnes	Average Au (g/t)	Cumulative Au (oz)
10	560,000	13.0	235,000
9	730,000	12.2	287,000
8	930,000	11.4	342,000
7	1,140,000	10.7	392,000
6	1,360,000	10.0	438,000
5	1,540,000	9.5	470,000
4	1,690,000	9.0	492,000
3	1,840,000	8.6	509,000
2	2,000,000	8.1	521,000
1	2,020,000	8.3	541,000

Table 12.4-2 Remaining unmined (via. open cut) resources by cut off grade. Rounding will cause summation errors.

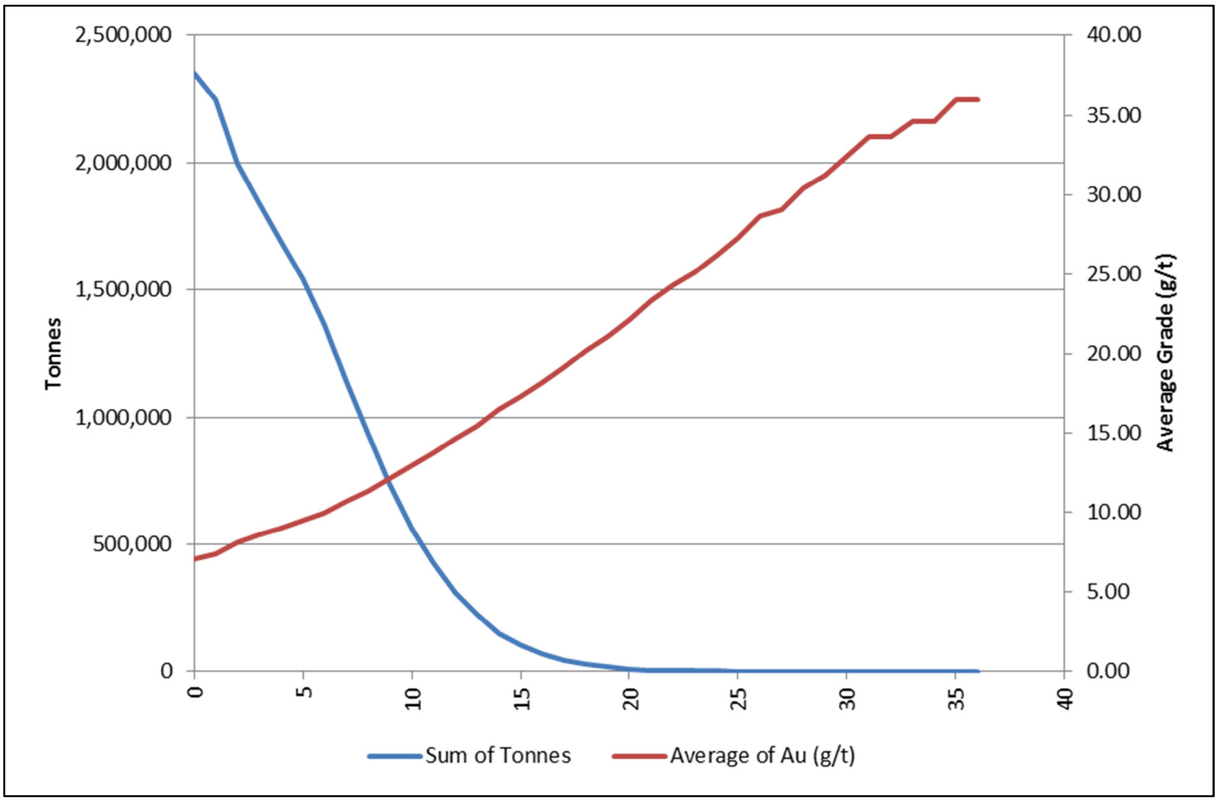


Figure 12.4-1 Grade tonnage curve of remaining resources after open cut mining.

13 Waste Storage Design

Constructed waste rock emplacements (waste dumps) are required to contain the waste rock generated by the mine.

13.1 Design Criteria

The following design criteria were used for the design of waste dumps.

Parameter	Value/Units
Bench height	10 meters
Berm width	5 Meters
Face angle	35 degrees
Rehabilitation Angle	25 degrees
Final Dump Swell Factor	1.3

Table 13.1-1 . Waste dump rock emplacement parameters.

13.2 Constructed Waste Rock Emplacement Designs

The constructed waste rock emplacements (CWRE) are shown in Figure 13.2-1. The volume of the dumps is shown in Table 13.2-1. The dump designs allow for a swell factor of 30%.

Dump	Volume	BCM	Tonnes
West	3,130,000	2,410,000	6,020,
Central West	5,210,000	4,010,000	10,030,000
Central East	2,830,000	2,170,000	5,430,000
East	8,450,000	6,500,000	16,250,000
Total	**19,620,000**	**15,090,000**	**37,730,000**

Table 13.2-1 Material quantities of waste dumps. Rounding will cause summation errors.

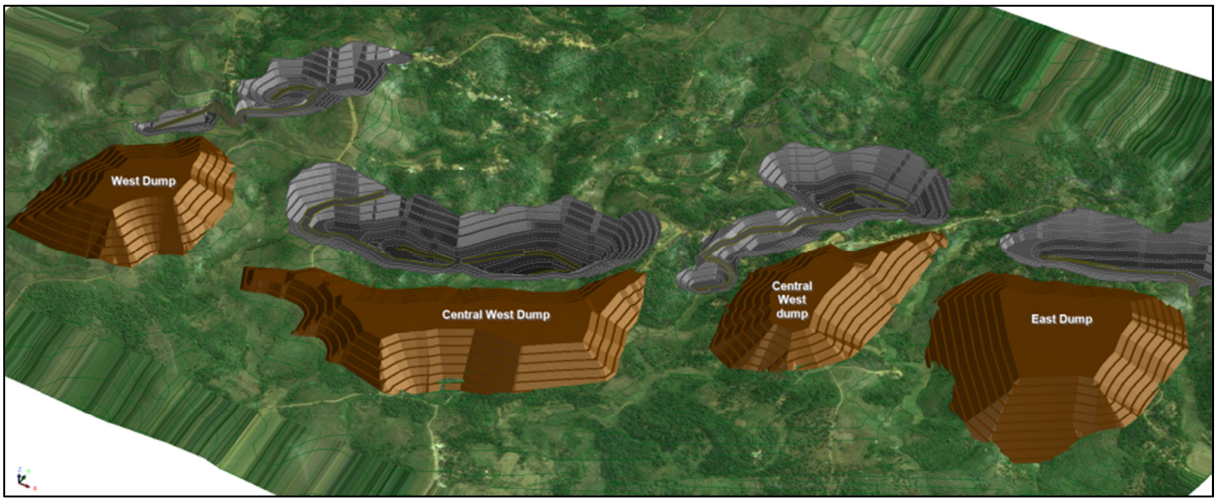


Figure 13.2-1 Isometric view of open cuts and waste dump designs.

13.3 PAF Storage

There is not sufficient information available on the existence or not of Potentially Acid Forming Waste. If this type of waste is encountered, it is recommended to backfill the PSM open cut followed by the SEK open cut when they are completed. This may require a rehandling of waste depending on the exact quantity and scheduling.

Storage of this material within the open cuts will also impact on the underground mining schedule and activities.

It is recommended that geochemical test work be undertaken to determine the PAF nature of the waste rocks and the results be assessed against the environmental management plan for the project.

13.4 Topsoil

Topsoil dumps have not been designed. Additional information is required to accurately calculate the depth of topsoil. It is recommended that the initial clearing and pre-production topsoil is stockpiled and used at the end of mine life for rehabilitation. Topsoil mined in subsequent years should be used for progressive dump rehabilitation each year.

14 Development

14.1 ROM

The Run of Mine (ROM) pad design will need to have a live capacity of 150,000 tonnes. Additionally, a low grade stockpile of some 230,000 tonne capacity is required to be located adjacent to the ROM. Detailed ROM and low grade stockpile design will need to be completed in future work.

14.2 Water Management

Effective water management will be crucial to developing the open cuts, meeting production targets and maintaining environmental compliance. Cut off drains directing water around the excavations will need to be constructed. Three major cuts through ridgelines are required to join catchment areas together and four dams are required to prevent water flowing into the open cuts.

Further detailed work is required in this area to confirm the size and width of these cuts. Further topographic information is essential to calculate catchment areas and local weather data for rainfall modelling.

The preliminary cut designs are shown in Figure 14.2-1 and globally in Figure Ex.1.1-1.

The process water dam will have variable decant ability and thus be able to draw water from the water diversion to its east during the dry season and spill back into the water diversion when full or during heavy rain in the wet season.

Additionally, each open cut will need to maintain a temporary sump with associated pumps that is carried down each progressive level at the excavation deepens. The sump/pump capacity will vary depending on the open cut exposure and the time of the year.

Feature	Cut (BCM)	Fill (m3)
Dam 1	0	40,000
Dam 2	0	10,000
Dam 3	0	10,000
Dam 4	0	10,000
Water Channel 1	90,000	0
Water Channel 2	90,000	0
Water Channel 3	150,000	0
Total	**340,000**	**50,000**

Table 14.2-1 Water management cut and fill quantities. Rounding will cause summation errors.

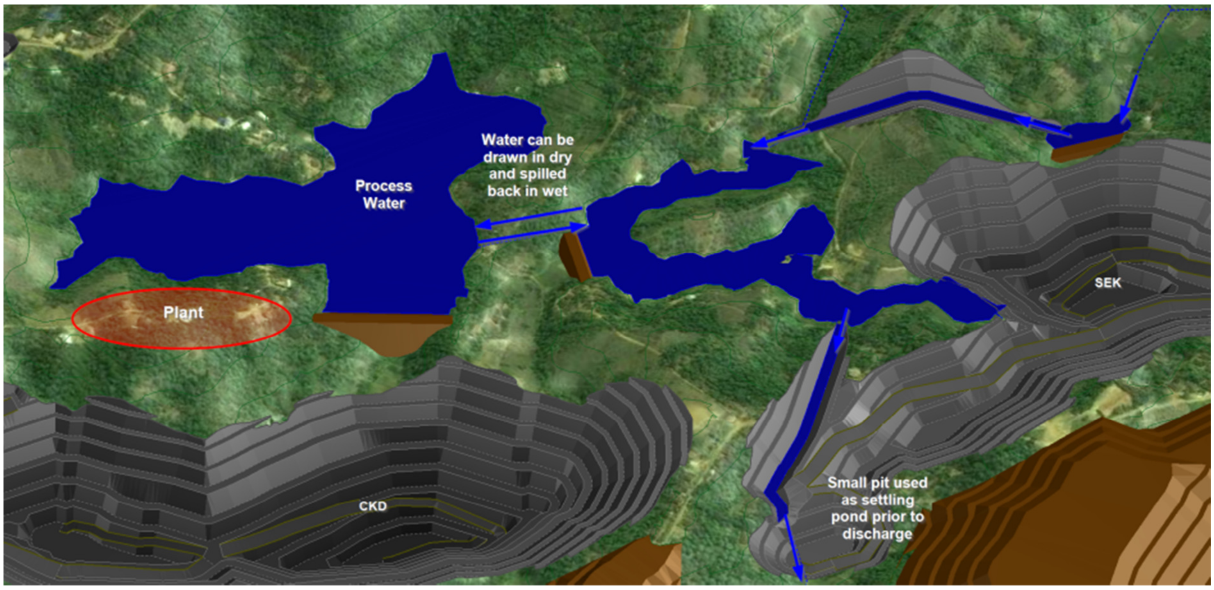


Figure 14.2-1 Isometric of conceptual water management features.

14.3 Roads

The operation requires a surface haul road to be constructed. The road can be constructed in stages as each new open cut commences. The physical quantities associated with the road are presented in Table 14.3-1. The road location is shown in Figure 14.3-1.

	Length (m)	Pavement Area (m^2)	Clearing Area (m^2)	Cut (m^3)	Fill (m^3)	Year of Construction
Road 1	3,000	60,000	66,000	94,000	254,000	Year 1-3

Table 14.3-1 Main haul road physicals.

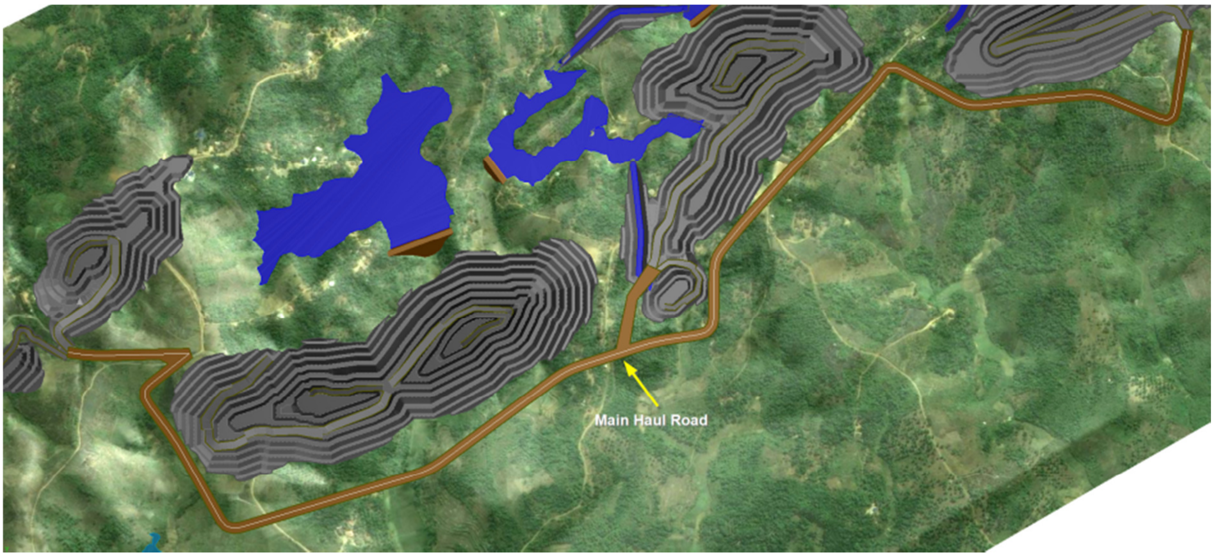


Figure 14.3-1 Conceptual main haul road route.

15 LAND DISTURBANCE

The total land disturbance from open cut mining will be 168 hectares. The land disturbance is shown by feature in Table 14.3-1.

Area of Disturbance	Quantity (hectares)
PSM	7.4
CKD	18.4
SEK	13.3
CBT	12.4
Total All Open Cuts	**51.5**
West Dump	14.2
Central West Dump	23.4
Central East Dump	13.8
East Dump	22.7
Total All Dumps	**74.1**
Tailings Storage Facility	10.0
Water Management	11.0
ROM & Plant	6.0
Roads and other	15.0
Total Infrastructure	**42.0**
TOTAL	**168.0**

Table 14.3-1 Land disturbance by feature.

Depending on current land ownership and whether appropriate compensation can be negotiated with the current land owner(s) land acquisition requirements are likely to be 2-3 times the land disturbance to provide buffer zones to the mining activity.

16 Mine Schedule

16.1 Production Schedule

The production schedule is based on the targeted productivity shown Table 16.1-1. The rise and fall in production is due to allowances for weather and working days per month. The schedule was designed to utilise two 90 tonne excavators and two 45 tonne excavators.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
90t Exc Kbcm/ Month	87	77	91	101	108	108	115	116	111	90	94	81	1178
45t Exc Kbcm/ month	36	32	37	41	44	44	47	47	45	37	38	33	482

Table 16.1-1 Monthly schedule targets.

The yearly production schedule is shown in Table 16.1-2 and Figure 16.1-1.

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Total
Movement (kBCM)	2,140	3,120	3,110	3,130	2,460	1,000	240	15,130
Waste (kt)	5,380	7,800	7,670	7,860	5,960	2,000	440	37,100
Ore (kt)	270	450	600	510	680	480	210	3,210
Au (g/t)	5.8	5.8	6.3	5.9	6.8	7.6	8.3	6.6
Au (koz)	50	83	124	97	149	118	56	677

Table 16.1-2 Yearly production physicals. Rounding will cause summation errors.

The mine is scheduled on 5m bench increments with each bench being completed prior to mining commencing on the next bench. Spread sheet files are supplied in Appendix E showing the schedule in monthly detail.

Based on the existing geotechnical data, open cut mining extends to a depth where fresh rock of moderate ground condition exists. Consequently, there is no need to extend the open cuts depth below that which generates the highest NPV for the purposes of establishing the underground workings. The need for such was postulated prior to the study commencing.

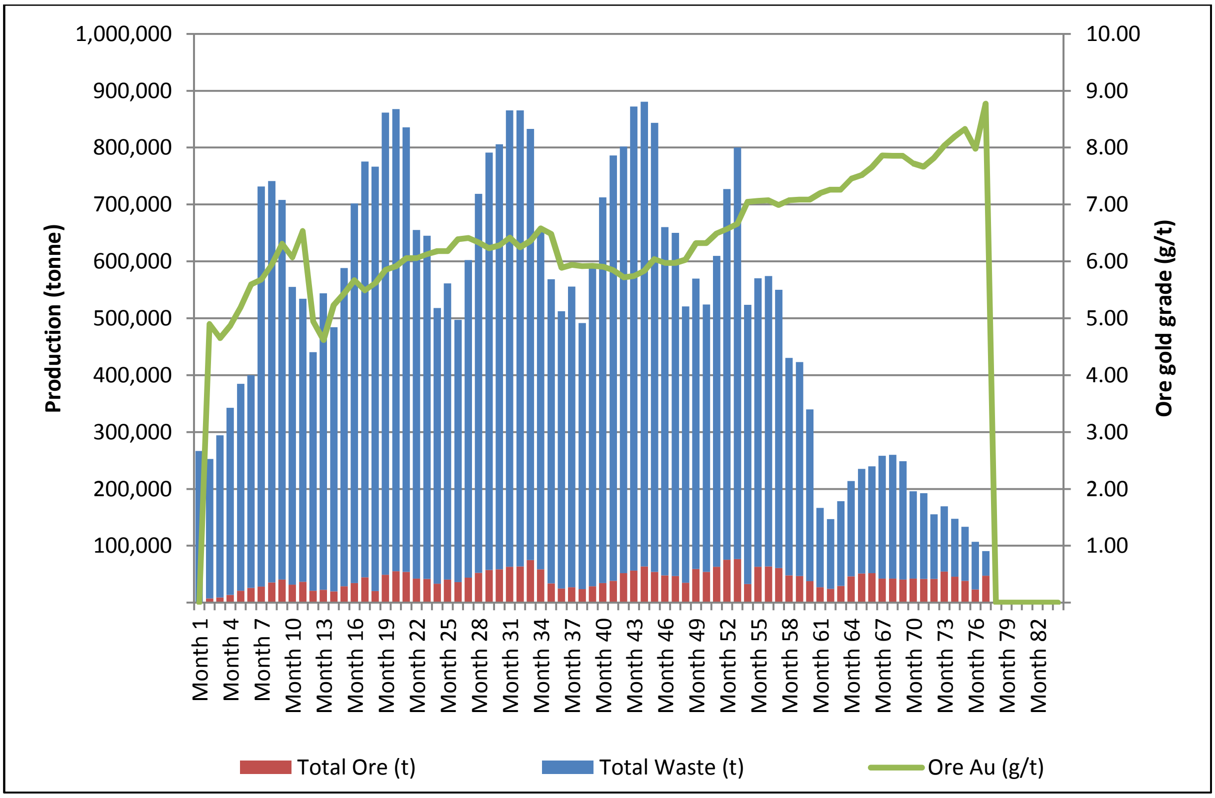


Figure 16.1-1 Monthly production chart.

16.2 Sub-grade Resources

The economic processing cut off grade has been estimates as being 1.3 g/t Au. Sub-grade (or low grade) ore, that being ore which has been mined but is less than 1.3g/t in grade will be placed on a stockpile adjacent to the ROM where is may be reclaimed if it can be economically processed and/or excess mill capacity exists.

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Total
LG (kt)	200	310	680	750	1,060	1,490	850	6,680
Au (g/t)	0.3	0.3	0.3	0.3	0.2	0.2	0.2	0.3
Au (k oz)	2	3	5	6	8	12	7	53

Table 16.2-1 Yearly production of sub-grade ore. Rounding will cause summation errors.

16.3 Stockpiling

In order to balance mine production with processing, a ROM stockpile will be used. The monthly stockpile size and grade is shown in Figure 16.3-1. The ore stockpile was targeted to not exceed 150,000 tonnes. Ore processing does not commence until the Month 10 of the mine plan. If ore processing is commenced earlier, there will be insufficient ore produced in some months to achieve the processing target of 1,500 tonnes per day.

In addition to ore stockpiling, sub-grade ore is also stockpiled for processing at the end of the mine life if economic.

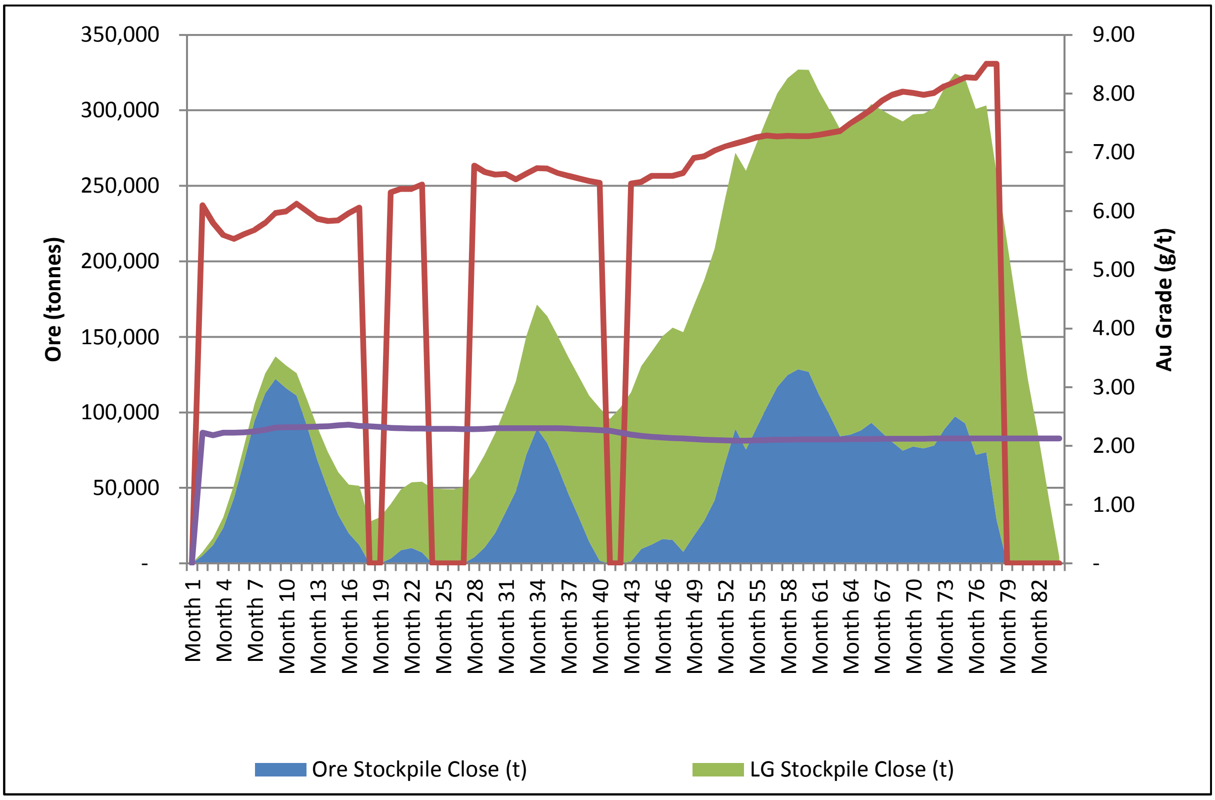


Figure 16.3-1 Stockpile chart.

More advanced scheduling using mine scheduling software and utilising a combination of drop cuts and/or delaying or accelerating ore from certain open cuts and benches would likely result in achieving an earlier process start date.

16.4 PROCESSING

Processing is based on 1,500 tonnes per day and the same estimated working days as the mining fleet shown in 15.4.1. A ramp up is included of:

- 500 tonnes per day in month 8
- 1,000 tonnes per day in month 9
- 1,500 tonne per day from month 10 onwards

During some months the direct feed ore production and ROM stockpiles are not adequate to meet the processing production target. In this situation, additional ore is drawn from the sub-grade ore stockpile to increase total processing to its target. The remaining sub-grade stockpile is processed at the end of the mine's life. A chart of monthly production is shown in Figure 16.4-1.

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Total
Ore Processed (t)	160,000	490,000	500,000	500,000	5010,000	510,000	280,000		2,940,000
Ore Au (g/t)	6.0	6.2	6.7	6.6	7.2	7.7	8.3		7.0
LG Processed (t)	0	20,000	10,000	10,000	0	0	230,000	10,000	270,000
Au (g/t)	0	2.3	2.3	2.3	0	0	2.1	2.1	2.2
Total Processed (t)	160,000	510,000	510,000	510,000	510,000	510,000	510,000	10,000	3,210,000
Average Au (g/t)	6.0	6.0	6.6	6.4	7.2	7.7	5.6	2.1	6.6
Total Recovered Au (oz)	28,000	89,000	97,000	95,000	106,000	113,000	82,000	1,000	609,000

Table 16.4-1 Annual processing physicals. Rounding will cause summation errors.

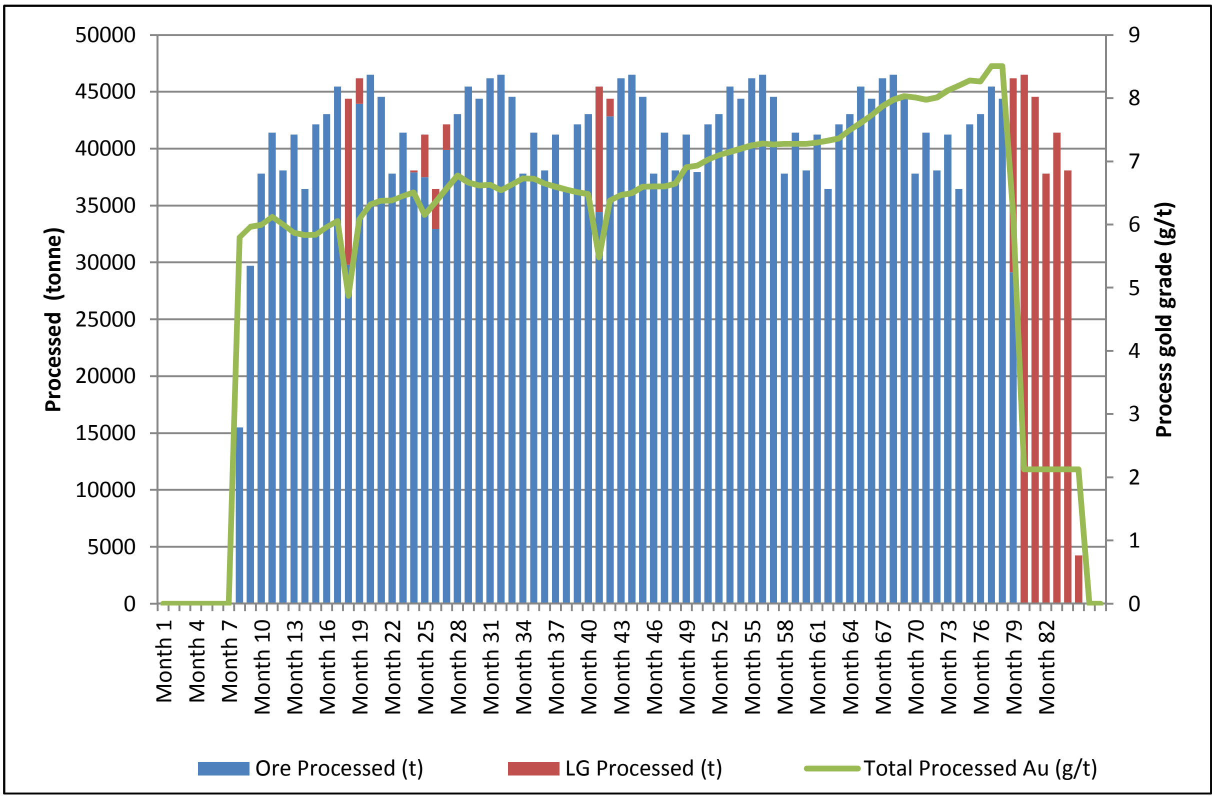


Figure 16.4-1 Monthly processing chart.

17 Mining Equipment and Productivity

The mining schedule is based on the following assumptions.

17.1 Drill and Blast

It was assessed that the first 35 meters from surface will be free dig. The material movement per month is presented in Table 17.1-1 and graphically in Figure 17.1-1 and Figure 17.1-2. Due to the relatively small proportion of drill and blast, contracting of this activity would be recommended. The drill rig should be suitable for conducting RC grade control drilling.

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Free Dig (BCM)	2,090,000	3,090,000	2,450,000	2,760,000	920,000	90,000	0	11,400,000
Drill and Blast (BCM)	60,000	30,000	660,000	370,000	1,540,000	830,000	240,000	3,730,000

Table 17.1-1 Annual free dig and drill and blast quantities. Rounding will cause summation errors.

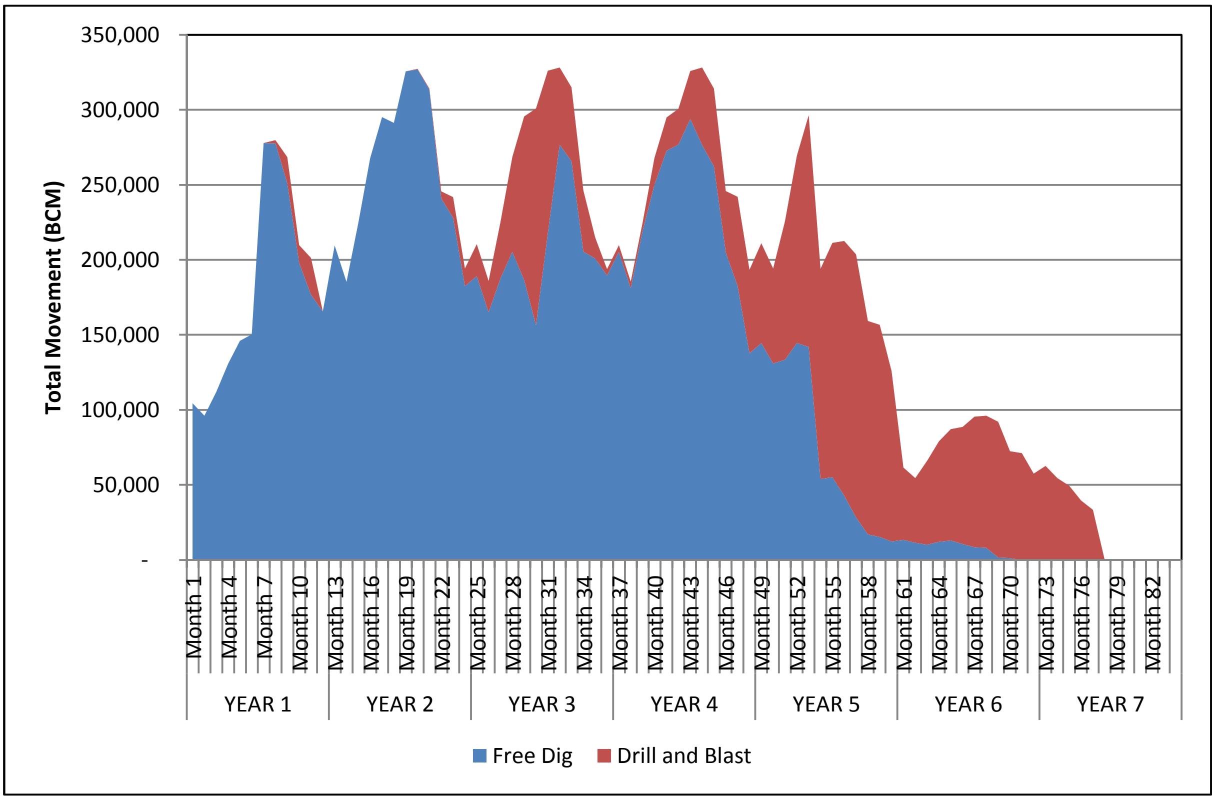


Figure 17.1-1 Material type movement per month.

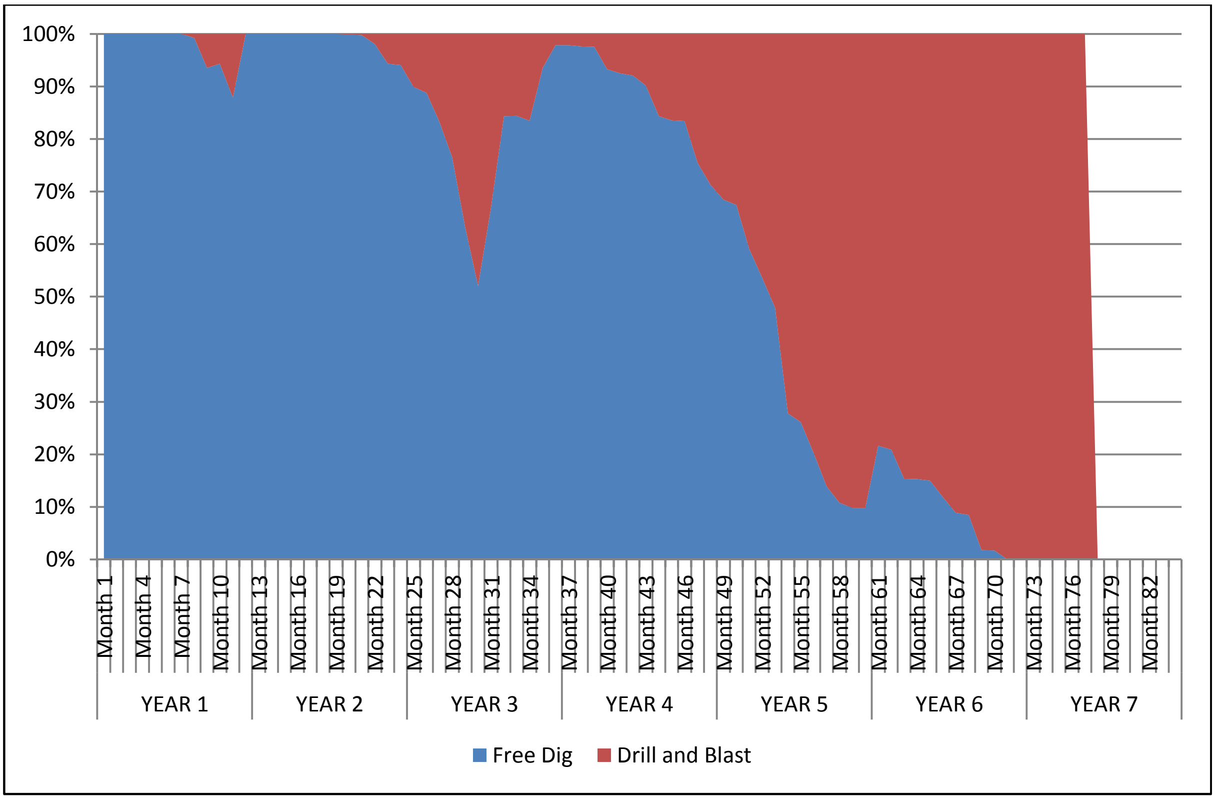


Figure 17.1-2 Material type proportion per month.

17.2 Excavation Fleet

The quantum of material movement by the excavation fleet has been based on the productivities shown in Table 17.2-1. The lost days allowance is based on regional public holidays and an allowance was made for a variation in productivity due to seasonal rainfall. This has reduced productivity by 15% during the peak of the wet season.

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Productivity Factor	0.85	0.85	0.87	0.94	0.96	0.98	1	1	1	0.96	0.91	0.85
Lost Days	3.5	3.7	2.9	1.3	0.7	0.4	0.2	0	0.3	5.8	2.4	5.6
90t Exc bcm/hr	178.5	178.5	182.7	197.4	201.6	205.8	210	210	210	201.6	191.1	178.5
45t Exc bcm/hr	76.5	76.5	78.3	84.6	86.4	88.2	90	90	90	86.4	81.9	76.5
Hours/shift	8.48	8.48	8.48	8.48	8.48	8.48	8.48	8.48	8.48	8.48	8.48	8.48
Shifts/day	2	2	2	2	2	2	2	2	2	2	2	2
Days/ Month	27.5	24.3	28.1	28.7	30.3	29.6	30.8	31	29.7	25.2	27.6	25.4
kBCM/ Month/ Exc	83	76	87	96	103	103	109	110	106	86	89	77

Table 17.2-1 Excavation productivity monthly variation.

The recommended mining fleet consists of two Cat 385 ME excavators mining waste only and two Cat 345 excavators mining a combination of ore and waste. These excavators are to load a fleet of Cat 740 articulated dump trucks. Articulated dump trucks were selected due to the climatic conditions and the expected soft ground conditions associated with a large amount of free dig clay rich material.

The mining schedule requires the excavation fleet throughout the mine life as depicted in Table 17.2-2. The Cat 740 (40t articulated dump truck) requirements are shown on a monthly basis in Figure 17.2-1. The truck requirement increase over the life of the mine as additional excavators commence and haul distances become longer due to deepening open cuts and distance to the plant.

Year	Excavator	No 40t ADT	Location	Comment
Year 0	90t Excavator 1	2	Water diversions	
	45t Excavator 3	2	Clearing and roads	Commences Month 6
Year 1	90t Excavator 1	3	PSM – SEK	Leaves PSM in Month 9
	90t Excavator 2	2	SEK	Commences Month 6
	45t Excavator 3	1	PSM –SEK	PSM Completed in Month 11
Year 2	90t Excavator 1	3	SEK	
	90t Excavator 2	3	CKD	Transfers from SEK to CKD in Month 13
	45t Excavator 3	1	SEK	
	45t Excavator 4	1	CKD	Commences in Month 13
Year 3	90t Excavator 1	3	SEK – CBT	Leaves SEK in Month 32
	90t Excavator 2	3	CKD	
	45t Excavator 3	2	SEK - CBT	Completes SEK in Month 35
	45t Excavator 4	2	CKD	
Year 4	90t Excavator 1	3	CBT	
	90t Excavator 2	3	CKD	
	45t Excavator 3	2	CBT	
	45t Excavator 4	2	CKD	
Year 5	90t Excavator 1	3	CBT	Leaves site in Month 53
	90t Excavator 2	3	CKD	Leaves Site in Month 60
	45t Excavator 3	1	CBT	
	45t Excavator 4	1	CKD	
Year 6	45t Excavator 3	2	CBT	
	45t Excavator 4	2	CKD	
Year 7	45t Excavator 3	2	CBT	Finishes Month 75
	45t Excavator 4	2	CKD	Finishes month 77

Table 17.2-2 Excavation fleet.

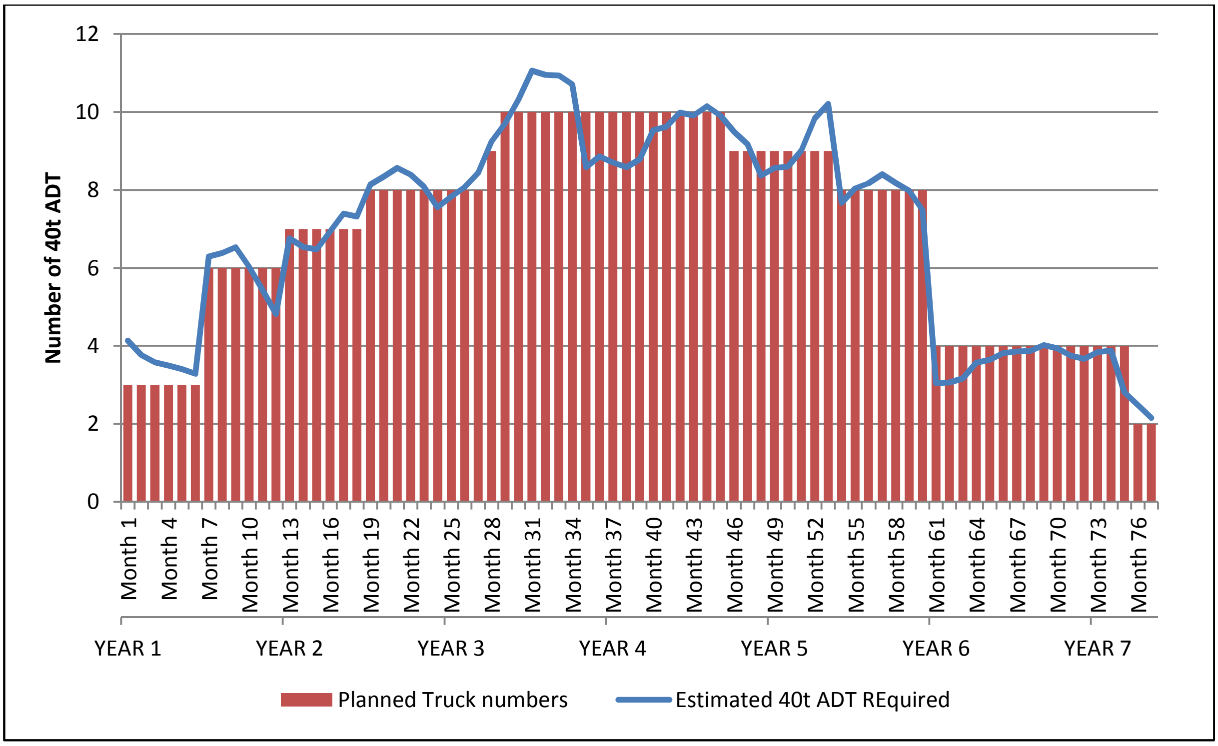


Figure 17.2-1 Haul truck requirements over project life.

17.2.1 Ancillary Equipment

The following ancillary equipment will be required to support the excavation fleet:

- Three Cat D9 dozers or equivalent. Two dozers will maintain two active dumps and the third will be used in the open cuts for ripping and clean up;
- One Cat 972 front end loader to manage the ROM and back up as a loading tool;
- One Grader for road maintenance;
- One TH340B tele-handler;
- One 30t excavator for on-going rehabilitation and water management;
- One spare ADT for rehab and infrastructure maintenance;
- Service truck;
- Water Cart; and
- Mobile pumps.

17.3 Grade Control Drilling

The quantity of grade control drilling using an RC drill rig has been estimated based on the scheduled ore tonnes. It was assumed that either a 10m or 20m or 30m hole will be drilled at 10m centres ever 2 benches (30m depth). This results in 35,000 meters of grade control drilling over the project life. Grade control was proportioned to each year based on the quantity of tonnes mined in that year. Grade control drilling is should lead mining by 3 months to allow for assay return and short term design and scheduling (Table 17.3-1). The grade control drilling will be carried out by a production drill rig with RC and sampling capability.

	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Total
Holes	150	250	350	300	340	300	100	1750
Drill (m)	3,000	5,000	7,000	6,000	7,000	5,000	2,000	35,000

Table 17.3-1 Annual grade control drilling requirements. Rounding will cause summation errors.

18 Cost Estimation

A high level financial model was built to evaluate the projects economics and to model project options. Detailed model outputs are presented in Appendix H.

18.1 Capital Costs

The non-mining related capital costs were taken from the estimates provided in the SRK Report. The capital estimate used in the financial model is shown in Table 18.1-1. The mining fleet capital cost estimate was updated to $20 million based on the preferred equipment selection and an assumed initial 12 month service supply agreement by the OEM. The pre-strip development capital has been classified as pre-production expenditure.

All capital has been depreciated to nil over the project life.

Item	Cost US ($,000))
Processing Plant	18,000
Tailings	9,000
Water	700
Power	1,600
Infrastructure	14,000
General	3,000
Others	13,600
Mining Fleet	20,000
Pre Strip	6,000
Total	**86,000**

Table 18.1-1 Open cut estimated capital costs. Rounding will cause summation errors.

18.2 Mine Operating Costs

The mining operating costs were estimated from benchmark figures of similar mines in Indonesia. The mine operating costs consist of the costs shown in Table 18.2-1.

	Cost per material tonne	Cost per month $,000
Loading & Haulage - Waste Mining	1.05	455
Loading & Haulage - Ore Mining	2.74	115
Drilling	0.20	26
Ancillary Mining Equipment	0.60	313
ROM Re-handle	0.14	5
Explosives	0.60	77
Diesel	0.43	23
Total Variable Costs	2.51	1,013
Total Fixed Monthly Costs	NA	606

Table 18.2-1 Mine operating cost estimate summary. Note: excludes pre-production expenditure. Rounding will cause summation errors.

18.3 Process Plant Operating Costs

The process plant operating costs from the SRK Report have been used as Mancala considers them reasonable and their use will allow meaningful comparisons of overall project economics of this study with prior estimates. In the financial model, total milling cost was estimated as $19.08 per tonne processed. Metallurgical recovery was 90% for all ore.

18.4 General and Administration

The general and administration costs were based on the SRK Report and bench mark figures. The average general and admin cost over the project life was $18.94 per tonne.

18.5 Selling Assumptions

Selling assumptions were based on Indonesian royalty rates and common refining charges. The gold price for the financial model was $1,300 per troy ounce. This was maintained for the entire project.

18.6 Taxes & Funding

The current financial model assumes a corporate tax rate of 25% and that all project funding is via equity and no debt funding is used.

Item	Cost
Indonesian Royalty Rate for Gold	3.75%
Dore purity	95%
Refining Cost 1 through put rate	0.29 per g of doré
Refining cost 2	$40 per kg of doré
Refining cost 3 Smelter retention	0.1% of contained gold
Smelter retention	0.1% of contained gold
Gold Price	$1,300 per oz

Table 18.6-1 Selling assumptions of gold doré.

18.7 Results

For financial modelling, project construction was assumed to commence in 2015. Mining was estimated to commence in 2016 with ore processing commencing the last quarter of 2016.

18.7.1 Costs

The average project cost per ounce of gold produced is shown Table 18.7-1.

Item	Units	Cost
Mining	USD/oz Au	228
Processing	USD/oz Au	101
General & Administration	USD/oz Au	100
Owners Costs	USD/oz Au	19
Refining Costs	USD/oz Au	3
C1 Cash Costs	**USD/oz Au**	**451**
C2 Cash Costs	**USD/oz Au**	**584**
C3 Cash Costs	**USD/oz Au**	**633**

Table 18.7-1 Total costs per ounce of gold produced.

18.7.2 Cash Flow

The project is forecast to produce approximately $301 million in positive cash flow over its life. The project will experience negative maximum negative cash in year two of approximately $77 million.

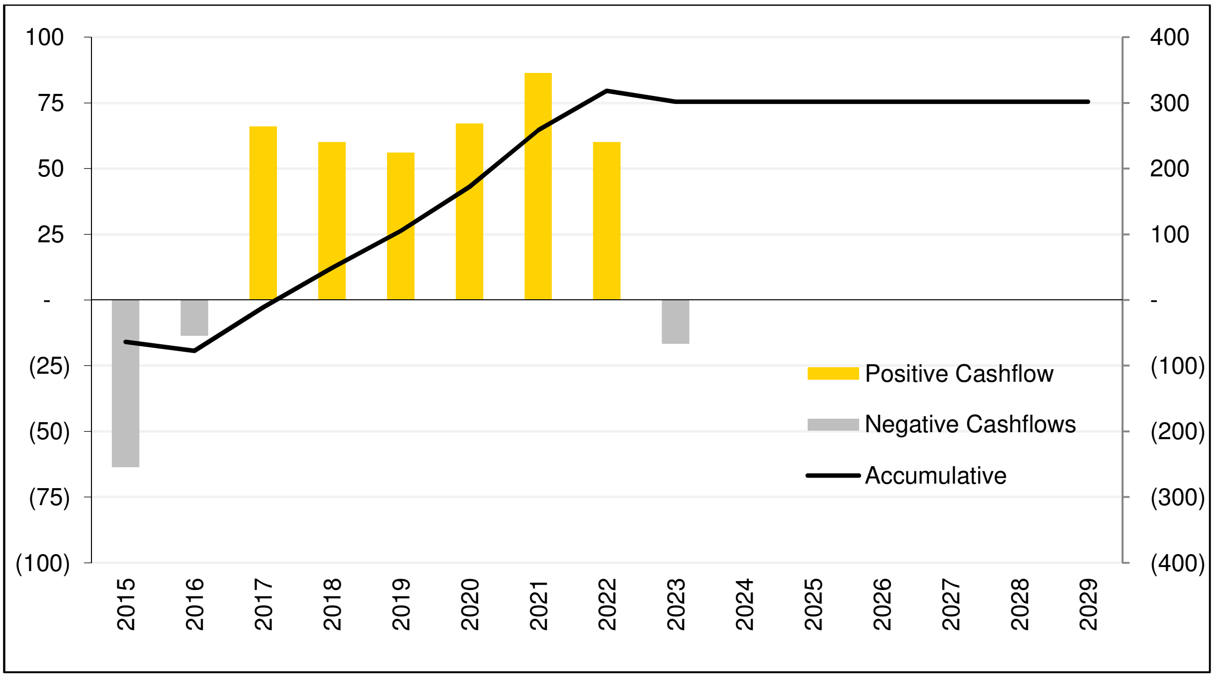


Figure 18.7-1 Estimated project cash flow.

18.7.3 Net Present Value

The Net Present Value (NPV) of the project at increasing discount rates is presented in Table 18.7-2. The NPV start date was assumed to be January 2015 when construction starts. The project's projected IRR is also shown.

Valuation	Unit	Project
NPV @ 3.00%	USD '000	251,000
NPV @ 4.00%	USD '000	237,000
NPV @ 5.00%	USD '000	223,000
NPV @ 6.00%	USD '000	210,000
NPV @ 7.00%	USD '000	200,000
NPV @ 8.00%	USD '000	186,000
NPV @ 9.00%	USD '000	175,000
NPV @ 10.00%	USD '000	166,000
NPV @ 11.00%	USD '000	156,000
IRR	% p.a.	53.%

Table 18.7-2 Forecast NPV and IRR.

18.8 Benchmarking

The Ciemas project and its development is relatively unique in present day Indonesia. The majority of existing or proposed surface gold mining operations are of considerably lower grade than Ciemas, which to achieve economies of scale, far higher production rates are required.

Perhaps the most comparable in terms of scale and the use of contemporary data is the 2014 publically published data from Sihayo Gold Limited for the Pungkut project in North Sumarata. Sumarata Gold and Copper also have a comparable scale project to Ciemas; however their published data incorporates costs associated with underground extraction methods, making direct comparison to surface operations difficult.

In relation to the non-mining relate capital costs as listed in SRK Report, Mancala opinions that the processing plant cost ($19M) is probably underestimated unless second hand plant items were considered in the cost. In comparison, Sumarata Gold and Copper are contemplating a 400ktpa CIL and Gravity plant costing some $21.8M after already expended some $20.0M at the site (largely on the processing plant). Table 18.8-1

The tailings water storage facility and the general infrastructure capital costs as listed by SRK ($9.0M and $14.0M) in Mancala's opinion are probably an overestimate. However, given the recently identified need for extensive water re-alignment channels and the geotechnical uncertainties as to the founding conditions for the TSF, Mancala considers that these estimate should remain until further cost data can be gathered.

The supply of reticulated mains electrical power to the site at a capacity suitable for the process plant and underground requirements for an estimated cost of $1.6M by SRK is considered an underestimate of what the actual cost will be.

Mancala's estimate of the mining capital cost of $26.6M could be considered high in comparison to the other operations. However the estimate it includes some $6.6M of pre-production costs and envisages a new tier one mobile plant fleet with an initial 12 months maintenance service and supply contract incorporated into the initial fleet purchase price.

In terms of operating costs, Mancala's estimate of $3.45/t material moved (ore and waste) is in alignment with other operations reported costs.

The SRK reported processing costs of $19.08/t ore are similar although slightly lower that other proposed operations.

The general and administration cost of $18.94/t as reported by the SRK report is considered high in comparison to the other operations. This would amount to some $800,000 per month over the project term. Realistic comparisons without the base detail (i.e. what is included in G&A) is difficult.

Mancala opinions at this level of study the capital and operating cost estimates are suitable and reliable. Further cost estimation work based on first principles is required to refine the projected costs.

Company	Sihayo	Sumatra Cu & Au	G Resources	Besra	Wilton
Deposit	**Pungkut**	**Tembang**	**Martabe**	**Bau Project**	**Ciemas**
Year Data	2014	2014	2009/2014	2014	2014
Type Data	BFS	BFS	BFS/Operating	BFS	Scope
Country	Indonesia	Indonesia	Indonesia	Malaysia	Indonesia
Commodity	Au	Au + Ag	Au + Ag	Au	Au
Processing Method	Gravity	Gravity + CIL	Gravity + CIL	Floatation	Gravity + CIL
TPA	750,000	400,000	4,500,000	2,900,000	500,000
Au Grade (resource)	2.4g/t	2.1g/t	1.9	1.7g/t	6.6g/t
Strip Ratio	3.4:1	11.0:1	0.7:1	1.6:1	11.6
Comment		U/G and O/Cut			
Capital Cost ($M)					
Owners Cost	5.7	5.5	66.4	0	13.6
Processing Plant	21.5	21.77	211	31.7	18.7
Infrastructure	25.5	3.52	0	16.4	27
EPCM	6.0	9.06	54.8	26.7	0
Sustaining	6.0	4.86	0	42.7	
Mining	8.1	7.41	10.3	24.3	26.6
Sunk Capital	0.0	20	0	0	0
Contingency	0.0	2.44	17.5	0	
Total ($M)	72.8	74.56	360	141.8	85.9
Comment		U/G capital cost excluded	Contract mining assumed	Owner Operator mining	
Operating Cost (C1)					
Mining Cost					
$/t Material	3.29	2.18		2.02	3.45
$/oz	271	244		125	175
Comment		Cost excludes mine services.			Includes capitalised pre-prod. expenditure
Processing Cost					
$/t Ore Processed	24.68	21.58		26.26	19.08
$/oz Ore Processed	449	266		665	100.6
$/t G&A	6.58	4.86		0.55	18.94
Total C1 Costs $/oz	**840**	**672**	**587**	**1,235**	**451**
Comment	Diesel power supply comprises $221/oz of OP costs	Exclusive of silver credits	Detail of operation cost unavailable		

Table 18.8-1 Benchmarking Table.

19 CONCEPTUAL UNDERGROUND MINING

Based upon the current resource model, at the conclusion of surface mining there will be some 2.0Mt of mineralisation remaining within the walls and floor of the open cuts (see Table 12.4-1). Based upon the assumptions of the pit optimisation process, further mining of the remaining resources by open cut methods decreases the project NPV. Underground mining must thus be considered.

19.1 PROJECT SCALE

PT WWI has expressed the desire for a production rate in the order of 450,000 tonnes per annum (1,500t/d) in the early stages this will be supplied by open cut mining. Underground production from multiple sources will be designed to meet this rate.

The ore zones can be up to 600 metres in length, an average width of four metres and a bulk density of 2.7t/m^3, this equates to a tonnes per vertical metre of 6,500 t/vm. However, the underground mineable (high grade portion) is likely to be in the order of 50 to 60% of this number, that is around 3,000 t/vm. Typical production rates from narrow vein mines of this tenor are around 250,000 tonnes per annum. To achieve the planned production rate at least two underground ore sources are required to be concurrently mined.

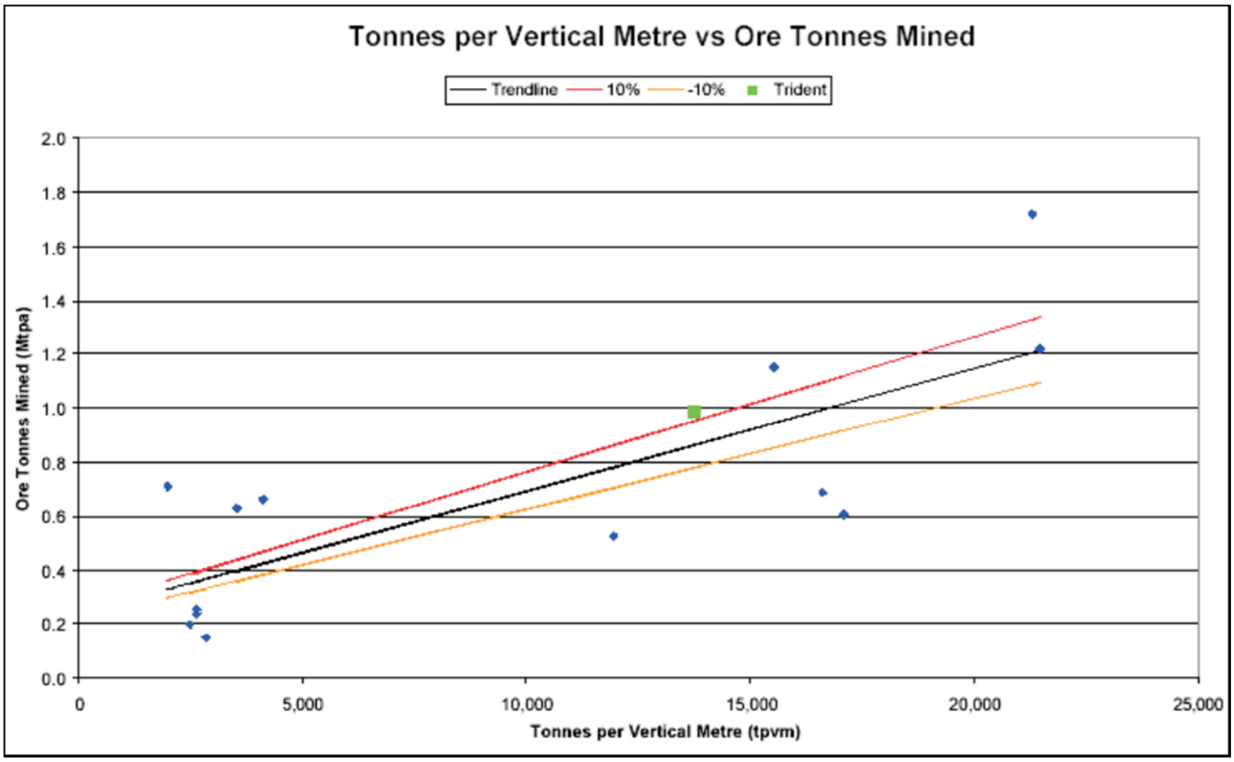


Figure 19.1-1 Production rate for narrow vein U/G mining vs. tonnes/ vertical metre.

19.2 Underground Mining General

The main portion of the know resources will be mined by open cut and the residual ore will be mined by underground methods. The mine design consists of a conceptual layout for the extraction of this ore. Access will be from the open cuts in the form of adits to mine residual ore in the open cut walls and from declines to mine the ore below the open cut floor.

Initially floor and wall pillars will be left to allow extraction but as mining retreats at the end of mine life these pillars can be partially extracted with retreat methods.

The open cuts will need to be designed to allow the underground ore to be accessed. This will require sufficient delineation drilling to establish economic and mineable underground blocks of ore. The open cuts will require detail design refinements to allow proper access to this ore.

Underground mining will need to be timed to provide continuity of production. Average decline development can usually achieve 50 to 70 vertical metres per year. Therefore it is possible to develop 3 production levels per year (15 vertical levels x 3). A further 6 months would be required to allow for the establishment of ventilation rises and escape man-ways. Hence underground development needs to start at least 18 months prior to the run-down of open cut production. If production is to come solely from underground sources two mines need to be developed concurrently. The alternate is to start earlier and develop the mines in sequence and have a combination of underground and open cut sources until two underground mines are fully developed.

19.3 Mining Methods

19.3.1 General

The ore-body and its surrounds have been classified by various geotechnical engineers as “moderate strength” except for local sheared zones of weakness (see SRK Report). The ore-body is relatively narrow varying from 1 meter or less to up to 10 metres. At Pasir Manggu it averages approximately 4 metres and strike lengths vary from 100 to 600 metres. From this it can be ascertained that mining will be predominately of the narrow vein style.

Applicable methods which are typically used in narrow vein mining are:

- Shrinkage;
- Mechanised cut and fill;
- Overhand benching with fill;
- Underhand benching with pillars and
- Alimak stoping.

These mining methods are discussed in more detail in Appendix F.

There is a trend is away from shrinkage to mechanised mining methods due to the low productivity and high costs inherent in shrinkage mining. Shrinkage methods typically yield 15 to 30 tonnes per man/shift, hence to achieve a production rate of 1500 tonnes per day 75 shrinkage miners are required. These numbers of skilled hand-held machine miners are unlikely to be readily available and therefore mechanised methods such as cut and fill and benching are preferred. Such methods are more productive, lower cost and significantly safer than hand held shrinkage methods. The main draw-back is the higher dilution inherent in these methods.

The proposed underground mining methods consist of:

- Residual ore below the floor of the open cut - Overhand Benching with Fill and
- Remnant ore adjacent to the sides of the open cut - Underhand Benching with Pillars.

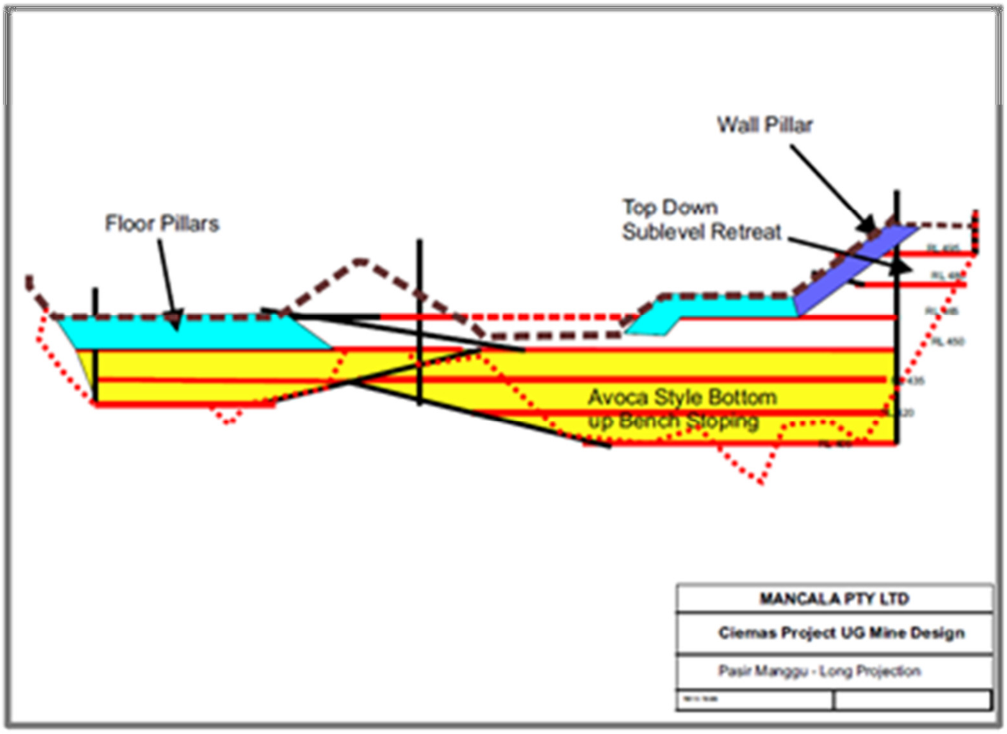
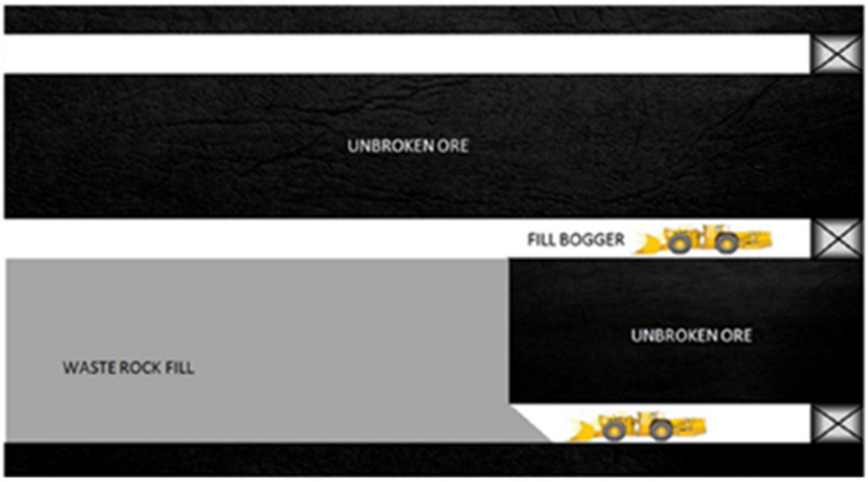


Figure 19.3-1 Illustration of possible layout to extract remnant ore from the open cut mine. (Pasir Manggu).

19.3.2 Residual Ore below the Floor of the Open Cut

The classification of the ore-body as “moderate strength” means the open spans will be limited and that pillars will be required to support the voids. As the target ore is high grade > 6 g/t methods that maximise the ore recovery are preferred hence fill methods such as overhand benching with fill (Avoca) are more attractive than underhand methods that require pillars.

The modified Avoca method is a bottom up style so it will be necessary to decline down to the base of the mineralisation or to a level where a sill pillar will be established. Mining will then occur in an upwards retreat style method until it reaches the open cut floor pillar. Floor pillars can be recovered at end of an individual mine life by a simple up-hole retreat method.



Schematic - Modified Avoca (reverse) stoping layout

Figure 19.3-2 Modified Avoca mining method.

19.3.3 Remnant ore adjacent to the sides of the open cut

This ore zones generally consists of limited length remnants in the walls of the open cut. Generally these remnants will be mined from adits from the side of the open cut. A wall pillar is required for overall stability and the ore beyond this pillar can be mined by up-hole retreat mining in a top down manner. Detail geotechnical investigations will be required to determine wall pillar dimensions and stable spans. The presence of local faults and zones of weakness will need to determined and incorporated into the design. These openings are generally small and as the ground conditions are considered moderate, the openings should be stable.

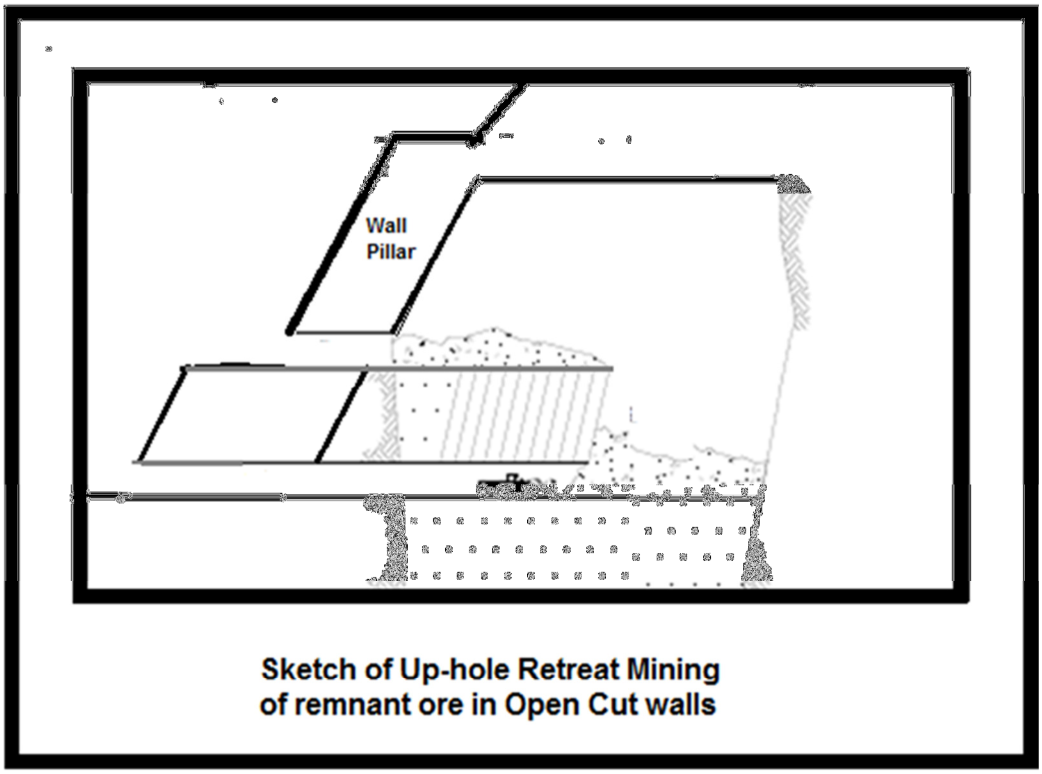


Figure 19.3-3 Up hole retreat mining.

19.4 Mine Development

It is envisaged that the declines will be approximately 5.0 x 4.5 metres in dimension and at a gradient of 1 in 7. Levels will be established at 15 metre vertical intervals, primary waste development will typically be 4 x 4 metres in cross section, with level ore development a minimum of 3 metres wide and 4 metres in height. This leaves 12 to 13 metres of ore (11 metres vertical) to be extracted between the levels.

This development combined with the modest production rate of 250,000 tpa from a single mine will allow the use of compact narrow vein type equipment namely 2 and 5 cubic metre loaders and 20 to 25 tonne trucks. Thus the development profile can be kept to a minimum.

19.5 Underground Capital and Operating Costs

No detail costing has been carried out therefore only indicative costs have been provided. These costs have been based on published data from similar operations in the South East Asia regions such as Gosowong, Tolukuma, Co-O Mine, Way Lingo and Ponkor/Cibaliung. No detail costs are available for Ponkor/Cibalung mines but published total operating cost of approximately US$900/ounce calculates to around $150/tonne for a 5 g/t head grade. This would tend to indicate that the mining cost is in the region of $60 to $80/tonne. In general the other mines have mining costs around $100 to $150/tonne when one allows for sustaining underground development.

19.5.1 Capital Development Costs

Each mine will require the establishment of approximately 3 levels below the floor of the open cut to sustain the planned production rate of 250,000 tonnes per annum. The design concept is to minimise development in waste and mainly develop the levels in ore. Capital development consists of the main decline development, main level accesses to the ore-zones and associated ventilation and escape-manway rises.

The cost of the initial development of an individual mine is estimated to be around $3.5 million.

Development Area	Metres	$/metre	Total $
Portal	15	10,000	150,000
Decline 1:7	350	4,000	1,400,000
Level – waste capital	200	3,500	700,000
Ventilation	50	3,000	150,000
Escape way	n/a	4,000	200,000
Pumps/Vent Fans etc.			900,000
Total			**3,500,000**

Table 19.5-1 Estimated capital costs associated with an individual mining operation. (Exclusive of capital associated with mobile mining equipment).

19.5.2 Underground Mining Costs (Operating & Sustaining)

Production will come from sources typically producing 250,000 tpa according to the attached cost graph (Figure 19.5-1) this would result in typical total mining cost (operating plus mine sustaining capital) of around $70 to $100 per tonne. As Indonesian labour cost are at the lower end of mining costs a cost of around $70 - 80 per tonne is be assumed to be appropriate.

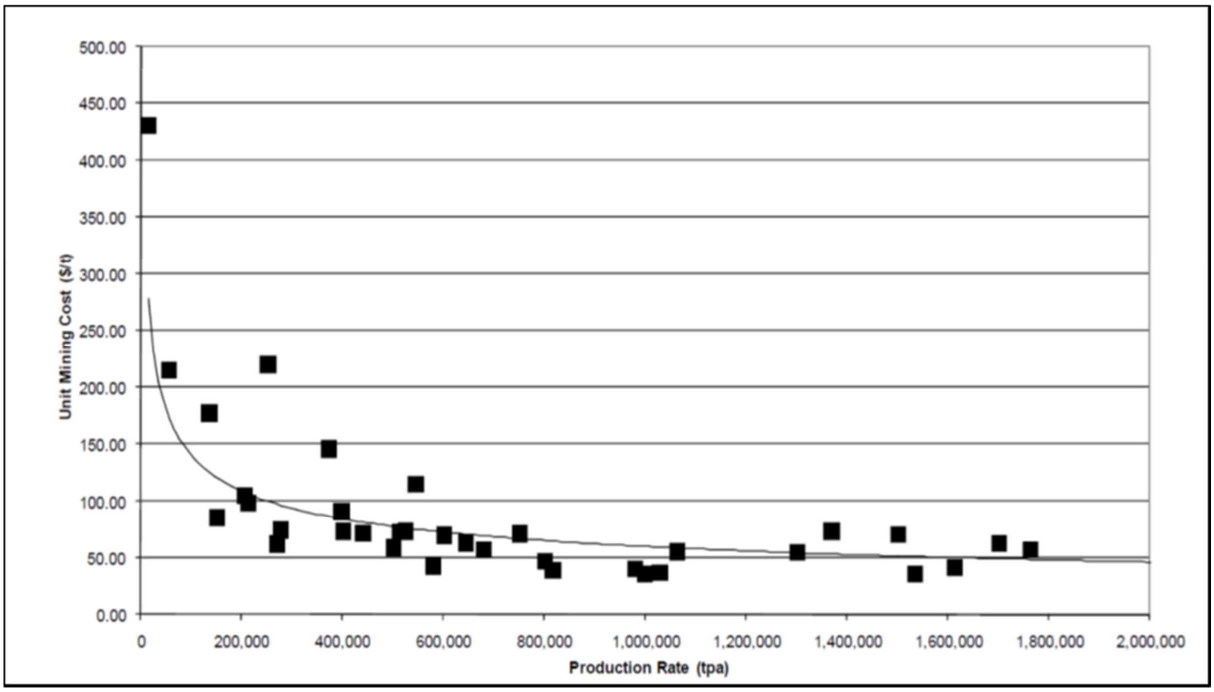


Figure 19.5-1 Typical underground mining costs vs. production rate.

19.6 Conceptual Mine Designs

Figure 19.6-1 to Figure 19.6-4 depict a conceptual mine design longitudinal projections for the four resource areas. Appendix I depicts a series of conceptual level plans for the development and mining of the Pasir Manggu post open cut mining.

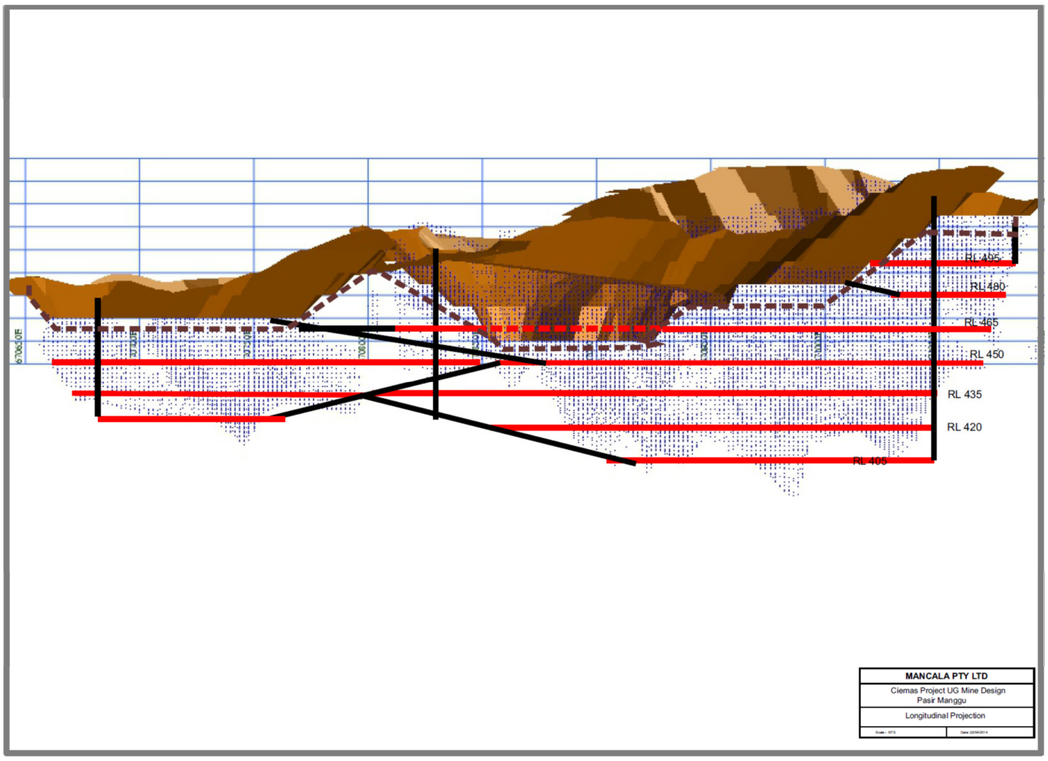


Figure 19.6-1 Longitudinal projection of conceptual mine design for Pasir Manggu post open cut mining.

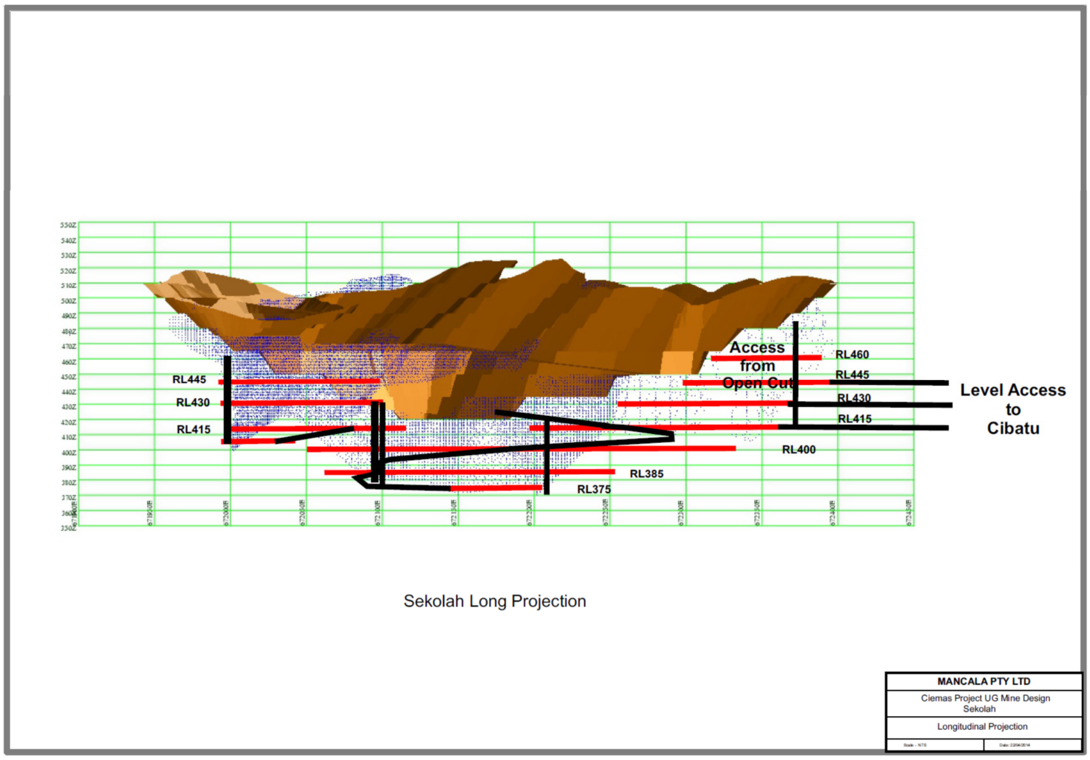


Figure 19.6-2 Longitudinal projection of conceptual mine design for Sekolah post open cut mining.

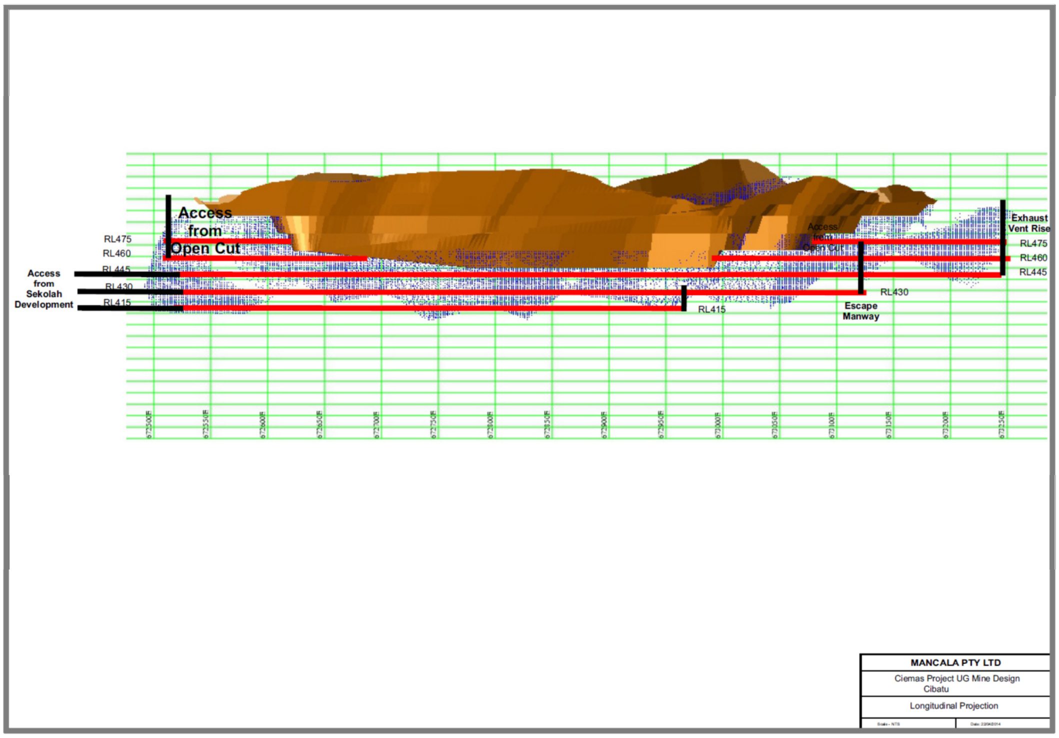


Figure 19.6-3 Longitudinal projection of conceptual mine design for Cibatu post open cut mining.

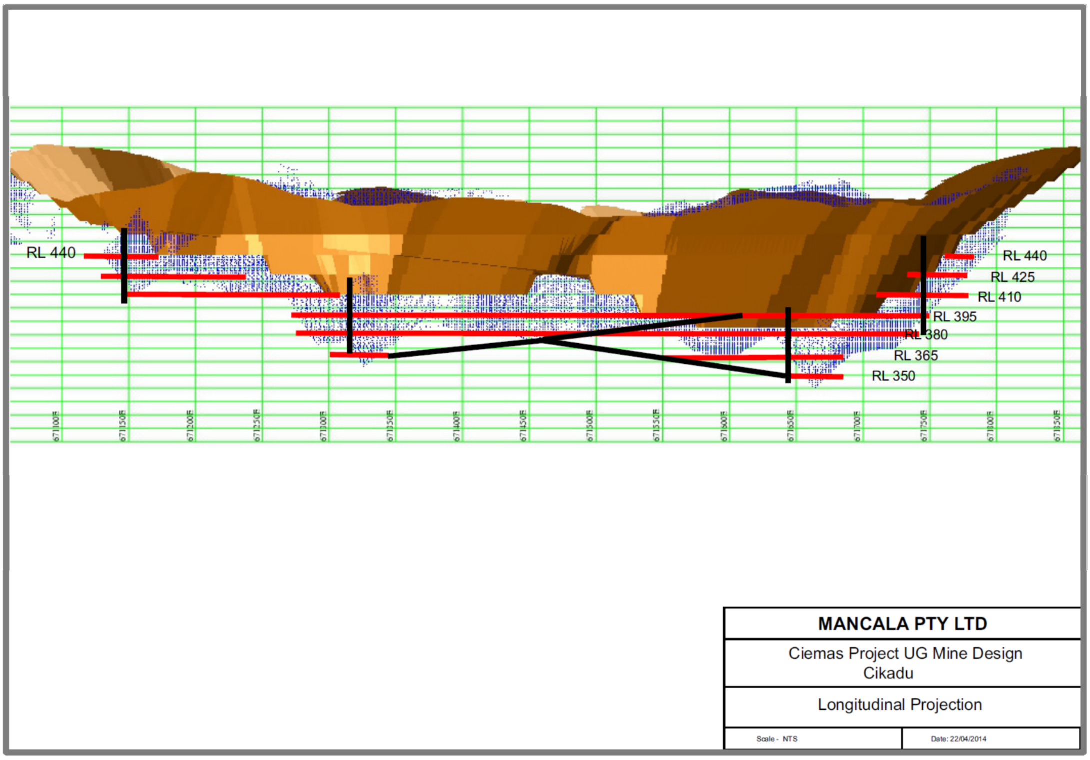


Figure 19.6-4 Longitudinal projection of conceptual mine design for Cikadu post open cut mining.

20 PROJECT INFRASTRUCTURE & ANCILLARY SERVICES

20.1 SITE ACCESS

Access to the site is provided by a paved roadway of some 45km length from the coastal city of Pelabuhan Ratu. Road conditions are fair although congested. From the paved roadway to the mine site, an unsealed road with poor alignment is present.

During construction large items of mobile plant and the componentry for the process plant will need to be transported to site. During operations regular deliveries of bulk goods (diesel fuel, explosives, mill reagents etc.) will be required.

The existing access road way to the site (from the west) will require upgrade to an all season duel carriage way roadway to accommodate regular bulk deliveries. The upgrade will require detailed design work in future studies.

20.2 SITE DEVELOPMENT & FACILITIES

The mine site is relatively isolated, currently supporting small scale agricultural activities. Development of the operation will require it to be largely self-supporting with the operation providing housing and messing for its employees and all technical and mechanical support facilities.

The surface mining operation will not require electrical power aside from de-watering infrastructure (in pit pumping/bore field?) and workshop infrastructure. The commencement of underground operations will see a significant power requirement for electro-hydraulic drill rigs, pumping, ventilation etc. Demand for a single underground operation will be in the order of 1-1.5MVA.

The processing infrastructure will require a significantly greater demand than the underground mines.

Although the capacity of the existing electrical power supply is uncertain, subjectively judging by the electrical conductor sizing it would appear not suitable for proposed milling and mining operations. Consequently, the electrical power supply will likely require upgrading, the design for which will need to be incorporated in future studies.

Other fixed infrastructures which will be required to support the mining and processing operation include:

- Mechanical workshops for both mine and mill with associated component storage and lay down areas;
- Fuel storage and dispensing facilities;
- Analytical laboratory;
- Core/sample storage and sample preparation facilities;
- Communication facilities;
- Training facility;
- Electrical distribution infrastructure;
- Messing and accommodation facilities;
- Emergency response and rescue facilities (including firefighting);
- Medical facilities;

- Security facilities;
- Explosive storage facilities (a magazine of limited capacity is present on site); and
- Technical, supervision and managerial office facilities.

The design and costing to supply such infrastructure to the project will need to be incorporated into future studies.

20.3 Surface De-Watering and Waterway Realignment

The Ciemas Project area is subject to some 4,000mm of annual rainfall concentrated within the months of November to April – a tropical monsoonal climate. Effective water management on the site will be critical for the project's success.

Incident rainfall on the open cuts will report to its base, from which it will require pumping to the surface. Sumps will need to be designed for each bench along with associated power supply and de-watering pipework.

The footprints of the proposed open cuts at Cikadu, Sekolah and Cibatu impinge on existing water courses. These water courses will either need to be re-directed or dammed to prevent surface water entering the workings. In addition, each open cut will require a diversion drain constructed around its outer limit to prevent inflow of minor surface flows.

Conceptually, two major waterway re-alignments are required along with four small impoundment structures.

20.3.1 Process Water Impoundment

It is proposed that an impoundment be created by building an earthen dam wall of some 10-15m height on the westernmost water course that crosses the Cikadu open cut. The water body formed would be used as the process water supply for the mill.

The western water course crossing the Cikadu would be similarly dammed, with impounded water directed westward to the proposed No. 1 waterway realignment channel. During the dry season, any additional process water requirements could be sourced from this water body.

The location and operational effectiveness of these impoundments are conceptual in nature and require detailed surface hydrological data to determine peak flow rates. Detailed topographical data, surface to sub-surface infiltration rates and site specific rainfall data will be required in future studies.

The presence of suitable material to construct the impoundments is uncertain. Test work to determine the location and quantity of material will be required. Dam design work should incorporate the risk of failure being as being significant (i.e. flooding the Cikadu open cut) and should be assessed with the consideration that the area is seismically active.

20.3.2 Waterway Realignment Channel No. 1

A water course transects the western end of the proposed Sekolah open cut. The flow rate and its frequency is uncertain until further studies are undertaken, however conceptually it could be re-directed westward. Such a re-direction would require a channel being cut of some 800m in length with a maximum depth from surface of 25m.

The channel would collect water from the north of Sekolah and flow would be re-directed from the second impoundment structure north of Cikadu. The channel could be directed into the completed southernmost Sekolah open cut to act as a settling dam.

The construction of the channel is a significant civil engineering exercise with a likely significant cost. Determining the seasonal flow regimes in the Ciemas water courses is essential to ensure the engineering exercise is appropriately scaled.

The channel's construction would be required during project establishment.

20.3.3 Waterway Realignment Channel No. 2

In a similar manner to the No. 1 channel, the No. 2 channel is proposed to re-direct surface water flow from entering the Cibatu open cut. From existing topographic data, the water course appears to be of greater significance (larger catchment area and a bridge crossing it to the south east) than other water courses on the site.

The proposed diversion channel would be some 500m in length with a maximum cut depth of 25m. A small impoundment may be required at the northern margin of the Cibatu open cut.

Construction of the No. 2 channel would not be required until year 4 in the project schedule, prior to the commencement of the Cibatu open cut..

20.4 Haul Road Construction and Road Base Quarry

Well constructed and maintained haul roads are an essential element of an open cut mining operation. The conceptual design has roads of 20m trafficable width and in total some 3,000m in length. Following cut and fill construction; the roads will need some 75cm of compacted sub base prior to topping with crushed road base. It is estimated that annually some 30cm additional road base will be required over the haul road surface.

During construction some 50,000 m^3 of road base material will be required for the major haul road. Additional material will be required for minor roadways estimated at 7,500 m^3. Ongoing road maintenance will require annually some 20,000 m^3.

Material suitable for road construction and maintenance is only known from one area at the Ciemas site, that being the location of the inclined shaft. A small quarry will need to be developed at this (or any alternate site) to provide in total some 160,000m^3 of material.

Suitable material may become available at depth within the open cuts. In this instance the quarry would become redundant.

20.5 Ore Stockpile/ROM Pad Preparation

The conceptual plant site (250m x 100m footprint) is located between the largest (Cikadu) and the earliest mined (Pasir Manggu) open cuts. The area is a ridgeline with moderate relief down to the proposed water process storage area.

An elevated ROM pad and ore stockpile area will be required at the southern end of the proposed plant site. The sites topography will allow the stockpile and the ROM to be partially cut into the hill side. Conceptually, an area of 80m x 220m is available for the ROM and stockpile combined. A good quality compacted base will be required for both the ROM and stockpile. Some 14,000 m^3 of material could be sourced from the road base quarry for this purpose.

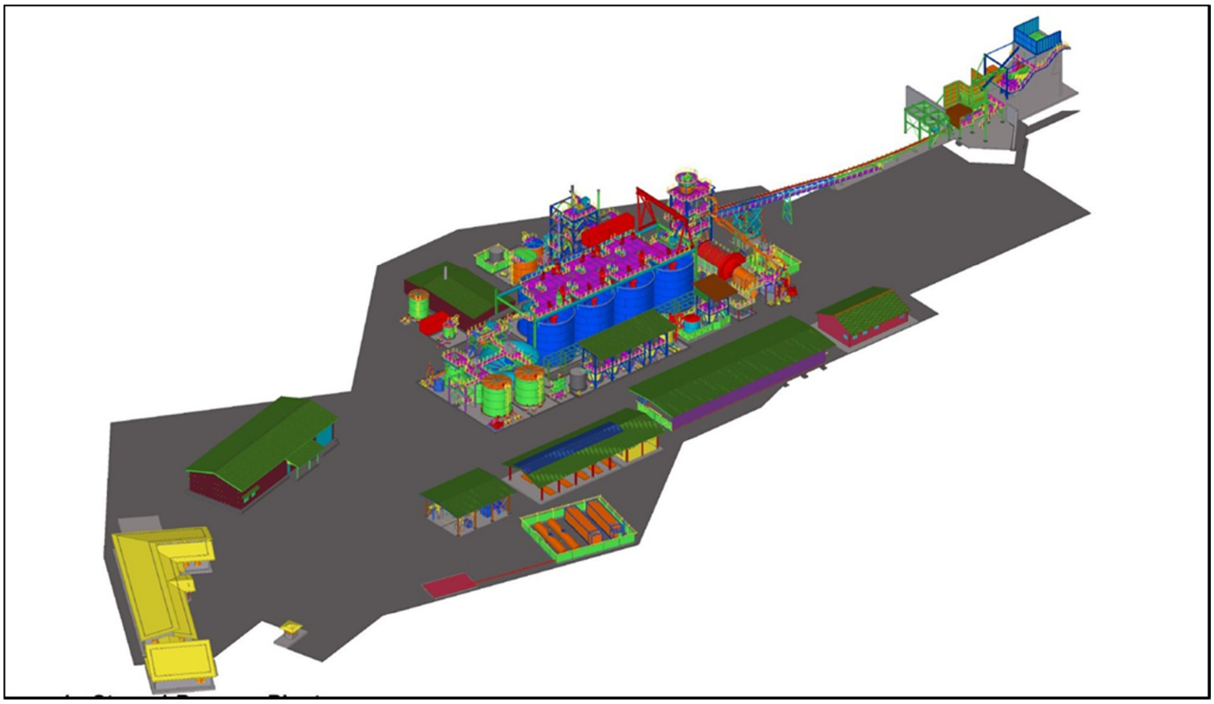


Figure 20.5-1 Typical arrangement of 400ktpa Au recovery plant. Single stage crushing, SAG and ball grinding, gravity separation, carbon in leach, AARL carbon striping, tails thickening and cyanide detoxification.

20.6 TAILINGS FACILITIES & MANAGEMENT

Some 5.5Mt of ore and dilution are planned to be mined by both open cut (3.2Mt) and underground methods (2.3Mt). Processing of these ore tonnages will result in approximately 3.6M m^3 of tailings which require long term stable storage. It is likely, due to the ore's sulphide content that subaqueous storage will be preferred.

A location for a tailing storage facility (TSF) has been identified in a valley south of the Pasir Manggu open cut. A dam wall of some 400m crest length and a maximum of 25m in height, impounding water to the 475m RL should provide adequate storage.

Detailed TSF storage design work is required incorporating:

- Review of other potential sites and selection criteria there of;
- Tailings quantity, size and density;
- Suitability of substrate below dam wall;
- Dam wall design considering seismicity;
- Construction in a number of lifts (delaying capital expenditure);
- Tailings dam closure strategy;
- The location and testing of suitable construction materials and
- Risk assessment incorporating identification of downstream infrastructure.

Clearly, the TSF is required prior to plant operations commencing.

20.7 EXPLOSIVE STORAGE

A significant quantity of overburden mined in the projects early years is potentially free dig. Material requiring drill and blast reaches a maximum of 1.5M BCM in year 5 with an average of 700k BCM over years 3 to 7.

Using a conservative powder factor of 0.3kg/t, on average the site will require 11 tonnes of explosive per week over years 3 to 7 and up to 23 tonnes/week in year 7. In year 1 and 2, approximately 1.0 tonne of explosives will be required on average per week. In addition explosives will be required for activities in the road base quarry.

A magazine of unknown capacity is currently located on site. Its capacity along with the estimated weekly consumption and ease of re-stocking should be considered in relation to future site storage capacity.

21 Ore Mining and Grade Control (Open Cuts)

21.1 Grade Control in Open Cut Mining

The mineral resources at Ciemas average 4.0m in width, are sub-vertical to 60 degrees from horizontal, are high grade and can be considered “narrow vein deposits”. To maximise profitability, the ore zones must be mined with minimum dilution and maximum recovery. To ensure these attributes are maintained, selective mining techniques are required.

The open cuts will be developed in either 10 or 15m final bench heights for the free dig or drill and blast zones respectively. The finial bench height will be extracted in either 2 or 3 interim benches of 5.0m height. Once blasted, swell will give an approximate 6.0m bench height which can be selectively excavated with two passes of a 45 tonne (ore) or 90 tonne (waste) scale excavators.

Grade control can be defined as the process whereby the mill feed is optimised in terms of maximising gold recovery from the material mined. This is achieved by minimizing dilution and ore loss at the production stage of a mining operation by the use of in pit sampling to define ore block boundaries. The grade control process is technically challenging requiring well planned and executed close spaced sampling programs using various sampling techniques.

Despite extensive sampling, ore loss and dilution will occur between the sample lines unless the excavation process is closely supervised by geological personnel. Potential sources of dilution and ore loss due to ‘edge effects’ is presented in Figure 21.1-1

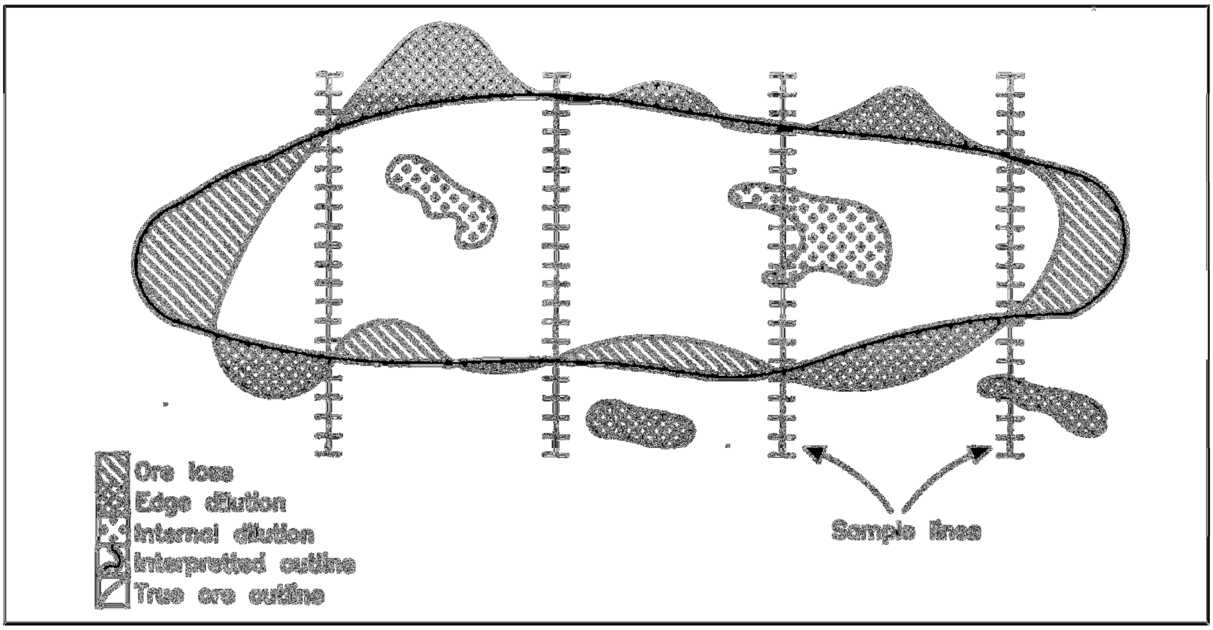


Figure 21.1-1 Edge effects producing ore loss and dilution on an open cut bench.

In general the grade control process consists of:

- Clearing the bench of all loose material;
- Close spacing sampling by either trenching, ditch witch or earth saw in free dig zones;
- Close spaced grade control drilling in the drill and blast zones;
- Geological mapping and chip sampling of the bench;

- Sample analysis and check sampling;
- Plotting of bench plans incorporating the geological mapping and sample results;
- Interpretation of ore and waste boundaries and estimation of flitch/bench in situ tonnage and grade;
- Ore boundaries marked and mining sequence planned;
- Mining by excavator under the supervision of a mine geologist, sending individual truckloads of ore to the appropriate stockpile;
- Stockpile management; and
- Reconciliation of bench/flitch tonnage to mill production.

The mining technique employed for ore/waste on individual flitches/benches will vary dependent upon ore body geometry, hanging and footwall ground conditions, ability to visually distinguish the ore/waste boundary and the presence or not of disseminated mineralisation beyond the ore body boundary.

The grade control process is invariably under pressure of the mining department to complete the exercise and commence mining. The major time constraint is assay turnaround. Samples need to be collected, prepared and analysed in as short a time (24-48 hours) as possible.

The recent advent of and held XRF tools can expedite the grade control process by providing in field assay results. Unfortunately gold cannot be analysed by XRF methods, and thus for its use in a gold setting, a positive relationship between gold and an indicator mineral is required. It was theorised arsenic may be such an element at Ciemas, but this does not appear to be the case (Section 6.8, Au vs. As Analytical Results).

Figure 21.1-2 displays various sampling techniques employed through the weathered zone of a typical near surface gold deposit.

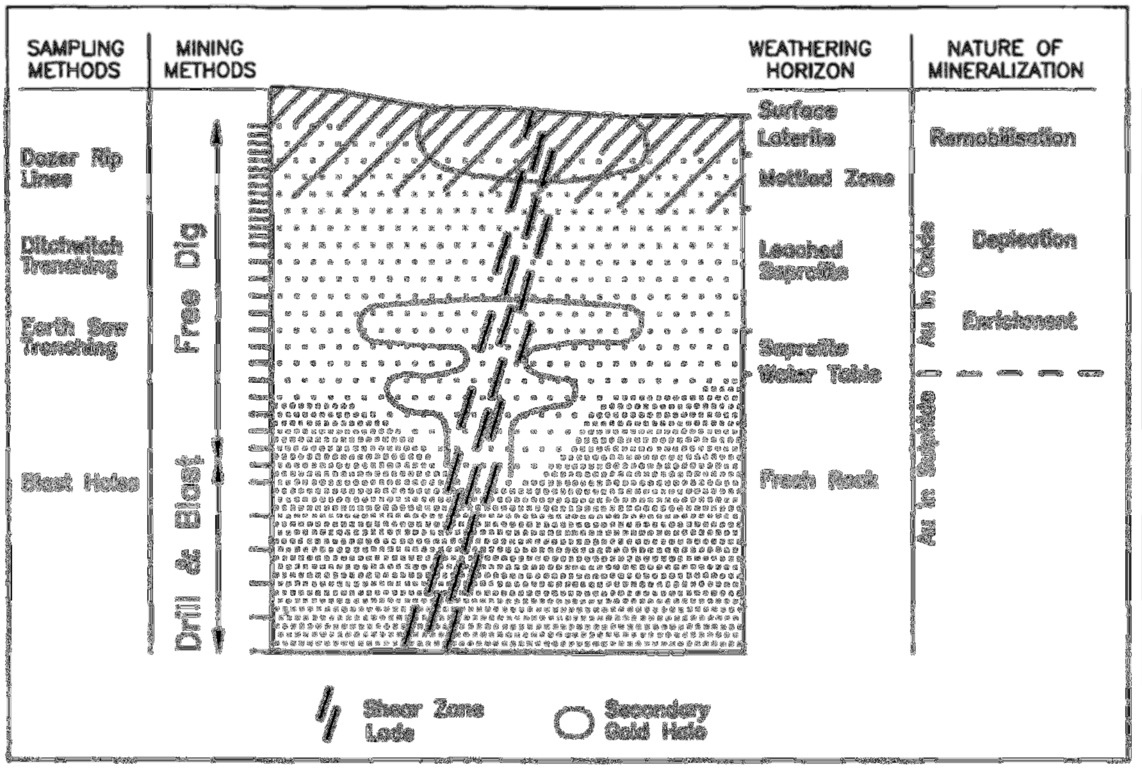


Figure 21.1-2 Sampling techniques for grade control purposes in the near surface region of a weathered gold deposit.

22 UNDERGROUND VS. OPEN CUT MINING METHODS

The SRK Report calculates a project NPV at a 10% discount rate using the mine design presented by Yantai. In its calculation, SKR has used various assumptions which differ to those of Mancala. To permit a realistic comparison of the two mining methods physical and financial outcomes and the perceived major risks, Mancala has partially modified the SRK modelling assumptions. Mancala has used these modified assumptions to estimate the financial outcome of the Yantai mine design and compared it to the open cut design (Table 22.1-1).

The most significant modification made by Mancala to the SRK/Yantai assumptions is to the estimated mining costs. SRK/Yantai estimate mining costs of $22.60/ore tonne. Based on bench marking and recent experience, Mancala estimate the mining cost for a mechanised, underground, narrow vein mining operation in Indonesia would be in the region of $80/ore tonne.

SRK/Yantai have assumed depreciation of the initial capital cost (USD 93M) at $7.107M/year, resulting in some $50M not being recouped over the project life. Mancala's open cut depreciation is also straight line, but is totally recouped over the project life. An adjustment to the SRK/Yantai costs has been estimated by Mancala to account for full capital payback over the project life (additional $8.3M/year in costs).

Mancala's NPV is reported after CIT at 25%. Mancala has made an adjustment to the SRK reported NPV to account for CIT.

SRK estimate of NPV is based upon a gold price of $1,400/oz while Mancala's work is based upon $1,300/oz. The SRK NPV has been adjusted by revising down revenue based on a gold price $1,300/oz.

The discount factor to NPV in Mancala's work is 8%, while the SRK model uses 10%. No adjustment has been made in this regard by Mancala.

Mancala's adjustment to the SRK financial model are estimates based upon the reported outcomes. Further accuracy would be gained if the assumptions were incorporated into the SRK base financial model. Mancala is not privy to this model.

22.1 Comparison between open cut and underground Mining Methods

Mine Physicals	Open Cut	Underground	Comment
Mine Life	7 Years	6 Years	3-4 years U/G mining after open cut
Production Profile	3.2 Mt 6.6g/t	2.4 Mt 7.1g/t	480k Au oz of Resource remaining after O/C
Gold (oz)	677,000	557,000	
Cut Off Grade	1.0	1.7	For O/C COG easily varied over mine life
Average mining dilution	16%	17%	
Ave. mining recovery	95%	85%	
Processing Rate (tpa)	500,000	450,000	
Processing Recovery	90%	90%	
Financial Outcome			
Pre-Production CAPEX	$86,000k	$93,000k	Owner operator O/C mining fleet, potential reduction if contractor option considered.
Pre-production schedule	1 Year	2 years	For O/C, further reduction in time frame possible with advanced scheduling
Operating Cost per ore tonne	$90	$123	For U/G, increase in mining cost of $57.20/ore tonne with respect to SRK Report. Based on benchmarking.
Gold Price (USD/oz)	$1,300	$1,300	Decreased from $1,400 for U/G
Project Cash Flow (EBITDA)	$488 Million	$315 Million	Decrease from $517M with adjusted Au price, increase mine costs and depreciation.
Discount rate	8%	10%	
NPV(8) (Post Tax Ungeared)	$186 Million	$120 Million	For U/G decrease from $210M with increase mining cost and post-tax.
Perceived Risk			
Availability of Miners	Low	Moderate - High	75 skilled miners required for U/G.
Availability of mining plant and serviceability	Low	Moderate - High	Remote area - limited OEM services available. Specialised U/G mining equipment.
Impact of poor ground	Low	High	Very poor ground conditions near surface.
Impact of water	Moderate	High	High rain fall and high water table.
Resource recovered	Low	Moderate	Pillars and unrecovered ore in U/G, suspected undefined resources recovered in O/C.
Surface impact	High	Low	Large footprint for waste dumps and open cuts.

Table 22.1-1 Comparison between open cut and underground mining methods.

23 Conclusions and Recommendations

The following primary conclusions were reached:

- Open cut mining of the deposit based on the optimised open cuts provides a better financial outcome compared with underground mining of the upper zones of the deposit;
- Adopting the open cut mining method increases the gold recovery per vertical meter as no pillars are left behind for support and adjacent un-minable mineralised lenses using underground methods will be recovered by the open cuts;
- Adverse ground conditions and ground water control are better managed by an open cuts compared with underground methods;
- Mining risks are significantly reduced using the open cut method; and
- Open cut mining significantly increases the area of land disturbance compared with underground mining.

Based on the primary conclusions the following primary recommendations are made:

- Development of the Ciemas Project should be changed to open cut mining of the upper zones of the deposits. The depth of the open cuts will be determined by optimisation techniques and comparative analysis of underground mining costs with waste stripping costs.
- Investigate the land usage and social impact for the open cut mining option.

Further conclusions and recommendations to particular areas are detailed in the sections below.

23.1 Geological Data and Interpretation

The geological data set for the Ciemas project has been accumulated over some 30 years by a number of owners. The exploration techniques employed, the data quality, quantity and reliability have varied considerably over time. As a consequence, a mixed data set is present, with most recent resource estimators excluding a significant portion of the data based on reliability concerns.

It is recommended that:

- An extensive search be made for original historical data (sample location, drill hole logs, assay data etc.). The surface costean data could easily be verified for location and potentially re-assay and the data included in future resource estimates;
- If historical data is not located, lithological and structural data could be obtained from the historical Parry drill sections;
- Known drill holes should be located on ground, their location checked and if possible down hole surveys undertaken;
- From existing down hole survey data, typical deflection curves could be generated and this data (where appropriate) could be applied to un-surveyed holes;
- Sterilisation drilling be conducted at the proposed location of all major infrastructure;
- Infill drilling is required on all resource areas to improve geological confidence and JORC status particularly for areas targeted for underground mining;

- A series of deep holes be drilled below the known resource areas to determine the potential scale of future underground operations;
- All future drilling to include silver, iron and other elements as directed by metallurgical professionals examining minor element impacts on the recovery process;
- Bulk density measurements be routinely collected for all future drilling and potentially existing core be re-sampled particularly in the oxide zone;
- Following the refined, verified and hopefully expanded data set, the resources should be re-modelled using lithological and structural constraints;
- The geotechnical data set be expanded by incorporating geotechnical logging and test work in all future holes. Near surface data in the free dig zone is particularly lacking. Geotechnical professionals should be directed toward providing recommended batter angles for the open cuts and stope spans for the underground;
- A LIDAR survey be flown over the entire area to obtain accurate surface elevation data and
- Acid/base accounting be conducted on waste rock to determine the PAF or acid neutralisation nature of the rocks.

23.2 Hydrological Data

Both surface and sub-surface hydrological data is very limited. The project is located in the tropical monsoon area which can experience torrential rain fall. Management of water on the site is critical.

It is recommended that:

- A catchment analysis combined with an infiltration study be undertaken to provide flow data on water courses which will be impacted by the open cut excavations. 1:100 year rainfall events impact on the surface infrastructure should be considered;
- Monitoring of water levels within existing bore holes should commence immediately and be conducted on a weekly basis over at least 18 months;
- Drawn down tests should be conducted on selected bore holes and
- Packer tests be conducted on selected existing bore holes, particularly where the holes intersect the mineralisation and/or major structures.

23.3 Infrastructure Design and Test Work

The major civil infrastructure works require detailed design work and to have test work carried out including:

- Determination of the quantity and suitability of site materials for impoundment construction (TSF and water course impoundments);
- Determination of the suitability of founding material for the impoundments; and
- Determine the quantity and suitability of material within the proposed road base quarry.

Infrastructure elements which require detailed and coordinated design include:

- Metallurgical test work, process plant design and construction;
- Site access and including light and heavy vehicle roadways;

- Tailings storage facility incorporating a staged construction, tailings analysis, impoundment wall design and closure management;
- Water way realignment channels incorporating surface hydrological data;
- Overall surface water management design and implementation including a site water balance;
- Electrical power requirements and reticulation at various stages of the project life;
- Explosive storage and management;
- Hydrocarbon storage and management;
- Location and construction of site buildings including workshops, technical/managerial offices, change rooms and crib rooms, the camp and messing facilities;
- The ROM and stockpile areas and
- Communication, security and medical facilities.

23.4 Pit Optimisation

Further pit optimisation is recommended once more detailed mining and processing parameters are known. Pit optimisation should also be completed after additional resource modelling and drilling.

23.5 Open Cut Design

Further refined open cut designs are required following new optimisations and for the feasibility design. The purpose of this updated design would be to achieve the following:

- Stage CKD open cut with at least one cut back and
- Reduce haul distances to waste dumps and improve integration of open cut exits with surface topography.

23.6 Scheduling

Detailed scheduling using a specific mine scheduling software is recommended as part of the detailed feasibility design. The purpose of this level of scheduling would be to achieve the following:

- Improve head grade in first 2 years;
- Level ore mining and potentially reduce stockpiling requirements;
- Bring processing start date forward;
- Provide increased reporting of more material types;
- Improve the dump utilisation and allow for PAF waste scheduling; and

23.7 Stockpiling

It is recommended that stockpiling be examined in detail to:

- Improve the NPV by increasing the high grade material processed earlier in the schedule and lower grade material deferred till the end of the schedule;
- Identify locations and costs of building and reclaiming from stockpiling locations;
- Allow mining to be undertaken in an orderly manner with as little inter pit movement of excavators as possible thus increasing the available work hours; and
- Enable processing to be levelled while mining retains the cyclical production schedule due to the wet season.

23.8 Mine Access

The open cut access and the Mine Access Road, ROM pad, go line and open cut accesses require better integration and optimisation.

23.9 PAF

There is currently not sufficient information known on the quantity and quality of potentially acid forming (PAF) waste. This is considered a significant risk and requires detailed design work in the next planning stage.

The analysis of PAF waste needs to be completed so that accurate volumes can be estimated. Once accurate volumes are known an encapsulation strategy needs to be developed that can contain the PAF waste.

23.10 Underground Mine Design

The conceptual underground mine design should be elevated to a Scoping Study level with capital, revenue and operating cost parameters incorporated into the overall financial model of the project.

23.11 Financial Model

The current financial model's cost estimates relies heavily upon benchmarked/industry average costs. Revenue similarly is estimated from limited metallurgical recovery data.

It is recommended a first principles cost estimate be conducted using supplier quotations for both operating and capital costs and productivity data matched to specific equipment.

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24.3 Acronyms and Glossary

ADT	Articulated Dump Truck
AMD	Acid Mine Drainage
BCM	Bank Cubic Metre
Berm	The flat step section of a pit design.
COG	Cut Off Grade
CWRE	Constructed waste rock emplacement
DFS	Definitive Feasibility Study
JORC	Joint Ore Reserves Committee
MBCM	Million Bank Cubic Metre
MI	Measured and Indicated Resources
MII	Measured Indicated and Inferred Resources
MT	Million Tonnes
NAF	Non-acid Forming
NAG	Non-acid generating
oz	Troy ounce
PAF	Potentially Acid Forming
ppm	parts per million (equal to grams per tonne)
RL	Relative level (from 0 which is mean sea level)
ROM	Run Of Mine
T	Tonnes

25 APPENDIX

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